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Final Report
TASK 1A - VBM Design Guidelines
Chapter 4

by

INCO Limited, Thompson Division
Thompson, Manitoba

December 1989

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RESEARCH & DEVELOPMENT IN MANITOBA MINES

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MINERAL DEVELOPMENT AGREEMENT
RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

Chapter 4

Drilling and Blasting

This chapter has been prepared as part of the

TASK 1A
VERTICAL BLOCK MINING DESIGN GUIDELINES PROJECT
of the
Canada/Manitoba Mineral Development Agreement

Prepared by

Golder Associates Ltd.
Sub-contracting to
INCO Limited, Thompson Division, Manitoba

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MINERAL DEVELOPMENT AGREEMENT

RESEARCH & DEVELOPMENT IN MANITOBA MINES

VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

The Vertical Block Mining (VBM) Design Guidelines consists seven chapters, published separately. The seven chapters are:

Summary

1. **Cut and Fill Mining**
2. **Data Collection Systems**
3. **VBM Stope Design**
4. **Drilling and Blasting**
5. **Support Requirements**
6. **VBM Stoping Procedures**

Reference to this chapter should be quoted as follows: "**Vertical Block Mining Design Guidelines Chapter 4 - Drilling and Blasting**"; Canada/Manitoba Mineral Development Sector B - Mining; DSS File No. 14SQ.23440-4-9147-3; CANMET Project No. 4-9147-3; December, 1989

CHAPTER 4

DRILLING AND BLASTING

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DRILLING AND BLASTING

1.0 INTRODUCTION

This report on drilling and blasting activities forms part of the main Task 1 (VBM Design guidelines). The objectives of the work are to:-

- describe these activities as they relate to the VBM system of mining practised at Thompson,
- compare these activities with like activities practised elsewhere (as considered appropriate) and
- assess the potential for improvements in VBM design and operating practices.

Production at the Thompson mine is at the rate of 6000 TPD, comprising about 80% from Vertical Block Mining (VBM) and the remainder from cut and fill stoping. VBM is a local term, derived from the concept of mining narrow orebodies in a sequence of "small" vertical blocks, using hydraulically placed backfill (Sarin, 1981). The drilling and blasting of the ore can be accomplished by any means, using small holes, large holes, crater blasting, slash blasting or slot/slash etc. Currently, nearly all of the ore is broken by crater blasting and a brief description of the technique is given in Appendix 4-A. Basically, VBM can be considered as a modification of the layout developed for mechanized cut and fill stoping; both methods are compatible, with comparable stoping advance per lift using crater blasting.

Stope blocks range in height to a maximum of 61 m (200 ft.) (Figure 4-1), using a standard size blasthole of 162 mm (6 in.) diameter. 115 mm (5 in.) diameter blastholes are used when sublevel interval is about 31 m (100 ft.) or less e.g. crown pillar recovery, blocks converted from cut and fill stoping to VBM (Figure 4-2), remnant recovery areas, etc. Overcuts and undercuts for longitudinal VBM are developed and/or slashed to generally a maximum width of 6 m (20 ft.) to minimize exposed backs during drilling. Hence blastholes are drilled in fans to intersect ore boundaries in lenses greater than about 4 m (12 ft.) thick and to envelop narrow, sinuous lenses (Figures 4-1 and 4-3). The general stoping procedure is to advance the back in increments of about 3 m (10 ft.) then empty the stope after each blast (Figure 3), using front end loaders equipped for remote operation. The final deck (crown pillar) is then mass blasted and drawn prior to filling the void with cemented sand fill.

Testing of the concepts of VBM commenced in December 1979 when four areas were selected for experimentation, with each of these representing a different mining application and geological setting (Sarin 1981). Since then, both productivity and profitability have been greatly improved by converting to this bulk method, especially in pillar areas (Stewart and Maclean, 1988). However, dilution of ore versus recovery of metal is a trade off. This is particularly so in high lift stoping and the mining of stressed remnants. Dilution in high lift stoping results from a number of factors, including the inability of long blastholes to envelop thin, sinuous orebodies without including waste and the sloughing of waste off the hangingwall. Loss of ore in stressed remnants can also be a problem due to "freezing" in deck blasts, inability to charge closed blastholes and the need to leave rib pillars for purposes of ground control.

In this report, drilling and blasting activities in current VBM are reviewed and compared with like activities elsewhere. Relevant data from this review, together with additional data collected for Sub-Tasks 4 and 5 respectively are then assessed as the basis for presenting blasting guidelines for future design. The need to monitor blasting and analyze the results are discussed and procedures to accomplish this are suggested. Finally, future needs and alternative approaches are presented for consideration.

The location of representative VBM case studies reviewed are shown in Figure 4-4. Reports and plans consulted are listed in the references and bibliography, but significant sources of data are listed below.

- 1401-301 Stope Stability Analyses (Longitudinal VBM above a cut and fill stope).
- 1803-330 Stope Stability Analyses (Transverse VBM above a cut and fill stope).
- 1803-331 Stope Stability Analyses (Longitudinal VBM above a cut and fill stope).
- 803-340 Longitudinal VBM in a narrow, flat dipping lens above a cut and fill stope.
- 3600L-231 Longitudinal VBM in a narrow to wide steeply dipping lens in 2 Shear. Stressed ground above a cut and fill stope.
- 1400L-310 Pillar Complex - longitudinal and transverse VBM (Ref. Stewart and Maclean 1988).
- Investigation of Blasting Practice dated April-May 1987. (Reference Golder Associates Report 872-1427).

2.0 CURRENT VBM PRACTICES

The technique of crater retreat stoping was developed from 1974 onwards at INCO'S Sudbury Basin mines in conjunction with Canadian Industries Limited (CIL). It was a response to the high cost of existing methods of pillar recovery and its use was extended to slot development for sublevel open stopes and as a replacement for cut and fill and post pillar mining. It is now favoured as a primary stoping method at both Sudbury and Thompson. Benchmark papers prepared by Lang, et al (1977 & 1978) described some of the work carried out at the INCO mines at Sudbury, the Birchtree mine near Thompson, Centennial Mine at Flin Flon, Heath Steele Mines at Newcastle, Sherrit Gordon Mines at Lyon Lake, Whitehorse Copper Mines at Whitehorse and Pamour Porcupine Mines, Porcupine. Other applications described were carried out at Rubiales, Spain and at Ardlethan, Australia.

Large scale VBM operations in deeper mining that can be usefully compared with Thompson are being carried out at the Homestake Mine at Lead, South Dakota, O'Okiep copper mine in South Africa and at Cobar Mines in Australia.

2.1. Thompson Practice

The majority of VBM stopes are converted from cut and fill stopes. Minimum stoping width is about 4 m (12 ft.) and stopes up to about 12 m (40 ft.) wide are mined longitudinally; wider stopes are mined transversely e.g. 1800L - 330 Block (Figure 4-6). Stopes wider than 8 m (25 ft.) are mucked through transverse drawpoints developed just below the back of the converted cut and fill stope (Figure 4-6). New VBM areas are usually developed with a footwall ramp to connect all sublevels;

Analysis of more than 120 mined blocks has provided the following rounded data on block geometry:

- Height range 15 m-61 m (50-200 ft.), Average height 40 m (130 ft.) ± 40 S.D.
- Width range 2-31 m (5-100 ft.), Average width 9 m (29 ft.) ± 18 S.D.
- Length range 6-40 m (20-130 ft.), Average length 18 m (60 ft.) ± 20 S.D.
- Dip range 45-90 deg., Average dip 70 deg. ± 10 S.D.

Other rounded statistics include:

- Rock type occurrences -

a) Hangingwall; schist 73%, quartzite 15%, peridotite 4% and other 8%.

b) Footwall; schist 46%, quartzite 34%, peridotite 16% and other 4%.

- Depth of hangingwall slough beyond planned stope limits; 46% none, 25% 0 m-1.5 m (0 ft.-5ft.), 27% 1.5 m-3 m (5 ft.-10 ft.) and 2% greater than 3 m (10 ft.)

- Depth below surface; 41% between 153 m-427 m (500 ft.-1400 ft.), 53% between 549 m-854 m (1800 ft.-2800 ft.) and 6% between 1,037 m-1,098 m (3400 ft.-3600 ft.)

- Tons per block; range 4,535-63,490 tonnes (5,000-70,000 tons), Average 18,140 tonnes (20,000 tons) ± 17,000 S.D. (drawpoint count).

- Distance hauled to pass; range 31-214 m (100-700 ft.), Average 104 m (340 ft.) ± 210 S.D.

- Tons mucked per shift; range 181-1,000 tonnes (200-1100 tons), Average 426 tonnes (470 tons) ± 240 S.D.

2.1.1 Drilling Practice

Raise and slot development is eliminated since the undercut provides the free face for blasting. The drill drive (overcut) is usually about 5 m (15 ft.) wide, hence holes are drilled in an inverted fan pattern, with toe spacing up to ten times that of collar spacing to cover stoping widths ranging from about 4 m (12 ft.) to 12 m (40 ft.). Fan drill patterns must also accommodate irregular ore/waste contacts, with drilling angles ranging from the vertical to approximately 55 deg. from horizontal. Ideally, all drill holes should breakthrough to the undercut, but in practice, many of the perimeter holes are "blind" and are fired initially as slash holes (Figures 4-1 & 4-3). Bunching of blastholes as stoping progresses from the undercut to the overcut is shown in Figure 4-7b to 4-7f. Blastholes are drilled with ITH (in the hole) hammer drills. Hole diameters between 114 mm (4.5 in.) and 165 mm (6.5 in.) are used in production stopes. Some statistics are as follows:

TABLE 4-1 DRILLING DATA

HOLE SIZE	TONS BROKEN DRILLED PER M	M DRILLED PER SHIFT
102 mm dia.	0.8 ± 0.9 S.D.	35 ± 23 S.D.
165 mm dia.	1 ± 1.9 S.D.	32 ± 32 S.D.

Details on hole burdens and spacings are discussed later.

2.1.2. Loading and Blasting Procedures

The first step is to inspect all of the holes and ream blocked or partly closed holes required for the blast. The second step is to profile both the back of the stope and the muckpile. These measurements are critical, since they determine charge placement, firing sequence (expansion room) and provide a record of breakage for each blast cycle (Appendix 4-A).

The charge configuration and geometry for 102 mm (4.0 in.) diameter and 165 mm (6.5 in.) diameter blastholes are given in Figures 4-8 and 4-9. Wooden blocks are lowered down the hole to a predetermined depth and tied off. Two or three capsules of stemming are then dropped to ensure that the hole is properly blocked.

ANFO and Aquamex (watergel) are the commonly used explosives. Standard charge weights are 2 x 12 kg (27.5 lb.) cartridges for 165 mm (6.5 in.) diameter holes and 2 or 3 x 2 kg (9.0 lb.) cartridges for 102 mm (4 in.) diameter holes. The explosive charge is centrally primed and initiated either by a down-the-hole delay system with a 450 gm slider primer and 5.3 gm/m detonating cord or collar delays with a 230 gm booster in association with 5.3 gm/m or 10.6 gm/m detonating cord. Detonators are short period nonels. Those parts of the explosive charge which enter the blasthole before and after the primer, are dropped from the collar to obtain maximum slump. Approximately 3 m (10 ft.) of stemming (usually mill tailing) is added to confine the charge.

Multiple charges in one blasthole are usually 1 m (3 ft.) long, spaced 3 m (10 ft.) centre to centre and separated by mill tailing. Deck charges are usually initiated by the down-the-hole delay system.

Statistics available for powder factor indices are presented in Table 4-2 below:

TABLE 4-2
POWDER FACTOR INDICES

HOLE SIZE DIA. MM	NUMBER VBM BLOCKS	POWDER FACTOR (KG/TONNE)	ROUNDED RANGE (KG/TONNE)
114	43	.5 ± 0.42 S.D.	.3 - 1
165	79	.5 ± 0.44 S.D.	.2 - 1

2.2 Case Studies Thompson Practice

2.2.1 1403-301 VBM Stoping Block

A comprehensive geotechnical/stoping performance analysis was carried out in the 1403-301 VBM stoping block during the period 1983-86 (Figure 4-4). The stope extends from 29,600N to 30,800N and is relatively tabular with an average mined width of about 2 m (17 ft.) and a dip of about 70° E (Figure 10). Design and production statistics are presented overleaf in Table 3. Mining commenced on 1400L as a cut and fill operation in 1970. In 1983, with 38 m (125 ft.) to be mined to 1000L, cut and fill mining ceased and VBM commenced. The original plan was to retreat mine from south to north, but mining commenced at each end, alternating VBM to provide continuous production. The last few central blocks were mucked from transverse drawpoints; the rest were mucked longitudinally. The mining sequence is shown in Figure 4-10. Blastholes were drilled at 114 mm (4.5 in.) diameter. Initial ore was mucked with a 2 m (2 yd.) scooptram and the remainder with a 3 m (3.5 yd.) scooptram.

Fourteen VBM blocks were mined, leaving four rib pillars for support (approximately 19% of the ore reserve). Detailed statistics are available for each VBM block and the data for all blocks are summarized below:

Rounded data on block geometry

- Ore width; range 2-7 m (6-23 ft.), Average width 3 m (11.3 ft.) ± 4.4 S.D.
- Mined width; range 3-7 m (11-24 ft.), Average mined width 5 m (17.2 ft.) ± 3.9 S.D.
- Average dilution by volume - 52%.
- Stope length range 12-29 m (40-95 ft.), Average 17 m (57 ft.) ± 16 S.D.

- Average dip of blocks 70^{o} E.

Rounded data on blasting

- Powder factor; ranges 0.3-0.6 kg/tonne (0.5-1.2 lbs./ton), Average 0.5 kg/tonne (0.9 lbs./ton) ± 0.3 S.D.
- Standoff (distance from back to centre of charge); range 0.9 m-1.4 m (3.0-4.7 ft.), Average 1.2 m (3.9 ft.) ± 0.6 S.D.
- Break per crater blast; range 1.25-2.5 m (4.1-8.3 ft.), Average 1.8 m (6.3 ft.) ± 1.5 S.D.
- Area of influence per 114 m (4.5 in.) diameter blasthole; range 1.2-3.2 sq. m (12-32 square ft.), Average 1.91 sq. m (19.1 square ft.) ± 6.4 S.D.

Rock Mass Quality Data

- The hangingwall in this stope was typically schistose in nature, the depth of the schist layer varying from 1.5-6 m (5 to 20 ft.) into the hangingwall.
- Rock Mass Quality of the hangingwall was fair (RQD >50%).
- The ore zone was slightly more competent than the hangingwall due to the lack of schistosity. However, a substantial number of shears and faults were found throughout the entire stope, primarily evident in the ore zone. These faults and shears most likely occurred in the hangingwall as well. However, access into the hangingwall rock was not available to substantiate this.
- Rock Mass Quality of the ore zone was good (RQD 75 per cent) despite these shear and fault zones.

The performance of individual VBM blocks is summarised below in the same order as the sequence of mining shown in Figure 4-10.

Block 1

Development drifts exposed the hangingwall schists in this narrow block and sloughing proceeded to the quartzite contact. The largest slough 25 m x 9 m x 2 m (80 ft. x 30 ft. x 8 ft.) occurred one month after the final blast; mucking ceased leaving 454 tonnes (500 tons) of ore. The broken ore supported the hangingwall during mining. Back stressing after mining as estimated by numerical modelling was 21 MPa; no drilling problems were reported (stuck rods). The average powder factor was 0.6 kg/tonne (1.13 lbs./ton).

Summary Excessive dilution resulted from a long hangingwall exposure 27 m (88 ft.) and heavy blasting.

Block 11

The strike length mined was 29 m (95 ft.). No drilling problems were encountered. Hangingwall and footwall rocks were relatively weak. The final bench blast was column loaded (some stripping holes were 31 m (100 ft.) long). 2,721 tonnes (3000 tons) of blasted ore could not be drawn. A low grade pillar was left between blocks 3 and 11 (Figure 4-10). Final caving outline is shown in Figure 4-11.

Summary Excessive dilution resulted from a long hangingwall exposure, weak stope boundaries and heavy blasting.

Block 2

The strike length mined was 15 m (50 ft.). No drilling problems were encountered. The average amount of explosive/delay was reduced by 50% compared with Block 11. Fifty-one misfires were reported and the average advance per crater blast was only 1.25 m (4.1 ft.), hence powder factor remained high at 0.6 kg/tonne (1.15 lbs./ton). Dilution was much reduced and there were few problems with ground control.

Summary Much less dilution with reduced strike length and less damage from blasting.

Block 4

The strike length was 14.7 m (48 ft.). No drilling problems were encountered. The standoff was increased from 0.9 m to 1.2 m (3 ft. to 4 ft.), increasing the average advance per crater blast to 1.7 m (5.3 ft.). There was very little overbreak, but dilution was heavy as the average ore width was only 1.8 m (5.8 ft.). A large portion of the bench did not break in the final deck blast.

Summary Good control of overbreak with a short strike length but performance was marred by failure of the deck blast.

Block 3

The strike length was 14.7 m (48 ft.). No drilling problems were encountered. The standoff was increased from 1.3 m (4.3 ft.) to 1.4 m (4.7 ft.), increasing the average advance per crater blast from 1.7 m (5.3 ft.) to 1.7 m (5.7 ft.). Ground conditions in the hangingwall and footwall were poor as indicated from lost core in exploratory diamond drilling and the condition of drift walls in the overcut. The average powder factor was 0.5 kg/tonne (1.0 lbs./ton) and 19 misfires were reported. Mucking was stopped during the annual shut-down and excessive sloughing off the hangingwall resulted in the loss of 454 tonnes (500 tons) of broken ore.

Summary Excessive hangingwall sloughing resulted from heavy blasting against a weak hangingwall (powder factor 1.0) and from the delay in emptying the stope after the final deck blast.

Blocks 6, 5 and 8

Generally good performances were obtained in these three blocks, with acceptable overbreak (except block 8) and low powder factors. No drilling problems were encountered. Ore skins were left in blocks 5 and 8 against the sand fill. There were signs of the extraction drift in Block 8 taking weight during mining. Induced stress for this block was calculated at 38 MPa.

Summary Generally satisfactory mining. First signs of stress induced failure noted.

Block 7

The strike length of this block was 20 m (64 ft.), but overbreak was similar to that of Block 8 (strike length of 15 m (48 ft.)). However, the hangingwall may have sloughed an average of 1.6 m (5.3 ft.) back to a shear. The slough temporarily buried an LHD. The blasting statistics were good, with a low powder factor of 0.53 kg/tonne (0.5 lbs./ton). There were only a few drilling problems. Estimated induced stress in the block was 27 MPa.

Summary Good blasting performance marred by a slough back to shear. The strike length of Block 7 relative to Block 8 was increased by 30% to 20 m (64 ft.).

Block 10

The strike length of this block was 16 m (51 ft.). Average overbreak at 2.7 m (8.8 ft.) was the largest in this stoping area. There were problems with stuck drill rods at the south end of the block; estimated induced stress in this area was 40 MPA. No drilling problems were encountered in the rest of the stope where estimated induced stress averaged 34 MPa.

Mucking was stopped when a large slough occurred 3 m x 5 m x 11 m (10 ft. x 15 ft. x 35 ft.) and 340 tonnes (375 tons) of ore was left in the stope. Blasting statistics were good, but powder factor at 0.81 lbs./ton was high considering the width of the stope.

Summary This is the first block where it is likely that a combination of blasting and stress induced failure contributed to overbreak.

Block 9

The strike length of this block was 18 m (59 ft.). Variability of ore width and narrower than expected ore width at the base of the block resulted in too few break through holes, hence the brow did not break

when blasted. Since the hangingwall schist was exposed, sloughing began and continued into the quartzite, with one wedge of 11 m x 5 m x 2 m (35 ft. x 15 ft. x 5 ft.) dimensions. The high powder factor of 0.475 kg/tonne (0.95 lbs./ton) probably contributed to the sloughing. 3,084 tonnes (3,400 tons) of ore were not recovered. No drilling problems were encountered.

Summary Poor back conditions and excessive sloughing of the hangingwall probably resulted from too few break through holes into the undercut and a high powder factor.

Block 13

The strike length of this block was 13 m (44 ft.). Blastholes in schist caved, but few drilling problems were encountered in solid ore. Estimated induced stress was 35 MPa. A drop raise was developed in the north end of the block and the remaining ore was slash blasted. A major back failure occurred in the extraction drift (12 m x 18 m x 2 m) (40 ft. x 6 ft. x 7 ft.). Average overbreak was 2 m (7 ft.) and powder factor was high at 0.44 kg/tonne (0.99 lbs./ton).

Summary Although no details are available on the charging configuration of slash holes, overbreak from some previous blocks is comparable. Induced stress was probably a contributor to the overbreak.

Block 12

The strike length of this block was 12 m (40 ft.). Estimated induced stress was 40 MPa, and extensive sloughing occurred in the drill drift. However, the block was in ore and no drilling problems were experienced. No figures of overbreak are available, but it must have been high because of the high dilution figure of 105%. Average break per crater blast was the highest for all blocks at 2.4 m (7.9 ft.) (stand off 1.2 m (3.8 ft.)). Average powder factor was only 0.23 kg/tonne (0.51 lbs./ton).

Summary A combination of stress and blast induced damage contributed to heavy dilution and probably large overbreak. Rib pillars were left either side of Block 14 to isolate completed Blocks 12 and 13 respectively (Figure 4-10).

Block 14

The plan was to mine the remaining ore in two stages. However, because of deteriorating ground conditions, it was mined between bounding rib pillars as one block, with a strike length of 17 m (55 ft.). Estimated induced stress was 46 MPa and the weak mineralized schist in the upper portion probably yielded; drilling and charging conditions were very poor. The drill drift, extraction drift and drawpoints deteriorated as mining advanced beyond the solid, bottom sulphides. After reconditioning, the remainder of the block was slashed. Powder factor was high at 0.6 kg/tonne (1.19 lbs./ton) but overbreak at 1.2 m (4.0 ft.) was surprisingly low. Hence it can be assumed that the overbreak figure given above refers to the part mined by crater retreat blasting only.

Summary Induced stress in weak mineralized schist estimated at 46 MPa was sufficient to cause considerable problems related to ground control, blasting and ore recovery.

2.2.2 800L - 340 VBM Stoping Block

This converted cut and fill stope is located in the upper part of 1 C ore zone between 600L and 800L and was mined over an 18 month period from July 1988 to December 1986. Five VBM blocks ranging in length from 12-20 m (40-65 ft.) were mined between elevations 10,000 and 10,100 (Figure 4), retreating N to S. The orebody is about 9 m (30 ft.) thick, dips 45 - 50 degrees E, and required 114 mm (4.5 in.) blastholes about 43 m (140 ft.) long.

The hangingwall was very strong and not heavily jointed; RQD was estimated at 100 per cent. The ore zone was jointed slightly more than the hangingwall, with an estimated RQD of 90.

The first three stopes were mined with full face cratering to the hangingwall contact without any significant overbreak. Block 4 was mined slot and slash with no overbreak. The bottom half of block 5 was mined with full face cratering and the top half was slashed against the sandfill in block 4. A large wedge sloughed off the hangingwall resulting in significant dilution. The RQD of the mined ore ranged from 25 - 59% over the five blocks.

Summary

Despite the flat dip of the orebody, VBM mining in this stope was successful. The thick ore and the low levels of stress were probably major factors in this regard. Slot and slash mining was successful in block 4 and it is interesting to speculate how this breaking sequence would have performed in block 5.

2.2.3 3600L - 231 VBM Stoping Block

This converted cut and fill stope is located in 2 shear ore zone between 3400L and 3600L and was mined over a 16 month period from May 1984 to August 1985. Four blocks, ranging in length from 18-24 m (60-80 ft.), were mined over an 24 m (80 ft.) interval with the overcuts located on 3400L. The 61 m (200 ft.) high interval above 3400L is unmined. Blocks numbered 1, 2, 3, 4 and 5 from S. to N. were mined in sequence 1, 5, 2 and 4, leaving block 3 as a 18 m (60 ft.) long rib pillar. The dip of the blocks was about 75 - 80 degrees with block 2 about 8 m (25 ft.) thick and the remainder 5-2.6 m (10- 12 ft.) thick.

Limited data was available for Rock Mass Quality determinations. Typically, the wall rocks in this area of the mine are heavily foliated, with joint spacings in the order of 2.5 cm (1 inch). The walls are near vertical, with little difference between the footwall and hangingwall. The wall rock is also more brittle in this section of the mine. As a result, a buckling type failure is not uncommon in the drill and mucking sills.

RQD's are hard to quantify for wall rocks due to the circumstances presented above. In general, the quality of the wall rock was poor, which would indicate an RQD rating of less than 50. The quality of the ore was better than that of the hangingwall, with an RQD of around 50 (poor to fair).

The wide 2 block was mined over a strike length of 34 m (110 ft.) with few problems in ground control and blasting. Blasting problems due to hole closure were experienced in the remaining 3 blocks, with frozen blasts occurring in blocks 1 and 4 which resulted in loss of ore. Slot and slash blasting as practised in the narrower blocks was not very successful, but minimal overbreak was reported.

Summary

Blasting problems due to hole closure resulted in loss of ore in the three narrow stopes, compared with no reported problems in the wider block 2. Overbreak did not appear to be a problem overall.

Compared with 800L - 340 VBM stoping block, the main differences are as follows:

- Mining was 793 m (2,600 ft.) deeper.
- About half the volume of ore was mined.
- The dip was relatively steep, but the RQD indicated weaker ore and wall rocks.

Hence it can be concluded that there is a potential for severe blasting problems associated with the mining of narrow ore.

2.2.4 1800L - 330 TRANSVERSE VBM Stoping Block

This converted cut and fill stope is located in the lower half of 1 C ore zone between 1600L and 2000L and was mined during the period 1985 - 1987. Eight 12.2 m (40 ft.) VBM blocks were laid out for mining between 1675L and 1850L (53.4 m) (175 ft.); ore width ranged from 24-42 m (80-140 ft.).

The rock qualities in this area of the mine are relatively high. The RQD's of both the hangingwall and the ore is in the order of 90. Some infilling of joints (carbonate and graphite infilling) was noted in these stopes, particularly in the hangingwall area.

Mining commenced in the thicker areas. VBM block 7 was mined and filled without incident. The adjacent block 6 was similarly mined in the following sequence:

- Mine a hangingwall slot with crater blasting and fill.
- Mine the remaining footwall section with crater blasting and fill.

The ground above 6 and 7 blocks caved 18 m (60 ft.) to 1600L after firing the deck blast in the footwall portion of 6 block and the void was filled within 6 months. The back of #7 block contained a large ball of peridotite, a rock unit which is notoriously unstable. The hanging portion of the adjacent block 8 was then mined and the cave expanded. A large portion of the fill was drawn with the ore.

Basically there were few problems with blasting but it is clear that recovery would have been improved with a good quality cemented fill, and the installation of higher quality support in the over-cut area. Back support installed was 24 m (8 ft.) mechanical bolts and screen with 12 m (40 ft.) cable bolts in blocks #6 and #7.

2.2.5 1800L - 331 VBM Stoping Block

This stope is located north of 330 transverse stoping block and separated by a significant transverse rock pillar. Four VBM blocks ranging in length from 12 - 20 m (40 - 65 ft.) were mined between 1675L and 1800L (125 ft.); ore width ranged from 3 m - 12 m (10 ft. - 40 ft.) and mining retreat was from N to S.

Problems were encountered with blastholes when mining was advanced 12 m (40 ft.) up in the first block and a deck blast was carried out when mining advanced 17 m (55 ft.) up because a lot of holes could not be cleaned. The stope was filled and a small pillar was left to separate Block 1 from Block 2.

Two shears were encountered at depths of 9 m (30 ft.) and 29 m (80 ft.) when drilling block 2 and blasting problems due to cracking and ravelling of blastholes occurred when mining had advanced up 9.1 m (30 ft.) above the undercut. Mining was stopped within 8 m (25 ft.) of the drill drift but the deck blast was not taken because the blastholes were crushed.

Some redrilling of holes was carried out for blasting in block 3.

Blasting problems in this stoping block resulted from a combination of stress and movement of blocks in sheared ground.

2.3 Summary and Conclusions

An average VBM block at Thompson only contains about 18,140 tonnes (20,000 tons) of ore. By industry standards, these are small tonnage blocks to apply VCR blasting techniques over high lifts (average 40 m (130 ft.)). Wall damage is inevitable with the high powder factors used (average 0.5 kg/tonne (1.0 lbs./ton)) to blast ore bounded by weak schists. The average depth of slough beyond planned stoping limits is 1.5-3 m (5-10 ft.) for 27% of the VBM stopes mined to date.

The average powder factor at Thompson is similar to the industry standard for breaking 3-3.66 m (10-12 ft.) slots in conventional longhole stoping. However, full column slash blasting applied to the average stoping width of 9 m (29 ft.) should permit powder factors of 0.25 kg/tonne (0.5 lb./ton) using ANFO and much less using diluted ANFO. Heavy blasting combined with higher ground stresses associated with remnant and deeper

mining only point to ever increasing dilution and ore loss with time, unless the amount of ore broken by VCR and associated deck blasting techniques is decreased appropriately.

The greatest challenge lies in reducing the dilution associated with the mining of narrow, sinuous ore bodies (less than 6 m (20 ft.) thick) while maintaining the productivity and costs currently being achieved. Case examples prepared by mine staff and described in sections 2.2.1 (1403 - 301 VBM stope) and 2.2.3 (3600L - 231 VBM stope) highlight the problems.

The hangingwall schist over the 366 m (1200 ft.) strike length of 1403 - 301 VBM stope can be described as fair (RQD greater than 50). Average ore width was 3.4 m (11.3 ft.) (range from 2 - 7.0 m (6 ft.-23 ft.)), but the mined width averaged 5.3 m (17.2 ft.), yielding an average dilution by volume of 52%. Additionally, 20-25% of the ore reserve within planned mining limits was lost in undrawn ore, frozen remnants from deck blasting and irrecoverable and fractured, rib pillars. The average data on blasting is given below:

- Blasthole size 114 mm (4.5 in.) diameter with area of influence, 1.97 sq. m (19.1 sq. ft.).
- Powder factor, 0.45 kg/tonne (0.9 lbs./ton).
- Break per crater blast, 1.92 m (6.3 ft.).

Difficult charging conditions were experienced in at least 5 blocks, at locations where estimated induced stress levels exceeded 35 MPA. Poor drilling and blasting conditions were experienced in the last two blocks mined at locations where estimated induced stress levels exceeded 45 MPA. Narrower than expected ore at the base of one block resulted in too few breakthrough holes. Very heavy blasting near the brow exposed schist, which then sloughed back to quartzite.

The rocks over the 153 m (500 ft.) strike length of 3600L - 231 VBM stope can be described as poor (RQD less than 50). Four stopes were mined over a height of about 24 m (80 ft.), with two stopes each side of a central 18 m (60 ft.) rib pillar. One of the stopes next to the rib pillar was about 6 m (20 ft.) thick, but the remainder were quite narrow at 3-3.6 m (10-12 ft.) thick. Drilling and blasting in the thick stope was problem free, but severe hole closure occurred in the narrow stopes, resulting in difficult charging, frozen blasts and lost ore. Only 50% of the holes were available for blasting in one stope. Compared with 1403 - 301 VBM stope, the rocks were of lesser quality, but mining was 671 m (2,200 ft.) deeper and induced stresses were high enough to close blastholes adjacent to narrow, 3-3.66 m (10-12 ft.) abutments. Hence there is a potential for severe problems associated with longhole blasting in deep, narrow ore.

Based on past experience, ground control problems are most severe when opening up a new block, but conditions tend to improve as the mining sequence is established. The reasons for these occurrences need to be understood. There is a possible need for improved ground support in the early stages of mining, but more importantly, enforcement of the minimum 3-2-3 pattern of breakthrough blastholes into the undercut may be

necessary if tone VCR blasting is to remain a major method of ore breaking in narrow orebodies. Changes to this pattern may be used if a modified blasting technique such as crater-slash is used.

Considering the mining of thick orebodies, there should be few problems associated with the closure of blastholes due to induced stresses. If problems do occur at some locations, they can probably be resolved by adopting a layout which involves the mining of an expansion slot followed by a mass blast of the remainder. An example of thick ore body stoping monitored by the mine staff is summarized in section 2.2.4 (1800L - 330 Transverse VBM stope). In this case, sufficient ground was undercut to cause back failure (possibly triggered by deck blasting practices). It seems that ground control would have been improved with good quality cemented fill (1 to 1.7 MPa (150-250 psi) uniaxial strength) combined with timely cable support and modified deck blasting practices. There were no reports of closure of blastholes due to induced stresses.

The cost advantages and higher mining efficiencies resulting from the single minded application of VCR blasting techniques have been shown to be offset in part by very heavy dilution in narrow, sinuous orebodies and reduced ore recovery, compared with traditional mining approaches. This is a conscious trade off, but it seems that the original concept of VBM as defined in the introduction is still appropriate, particularly the observation relating to the application of drilling and blasting practices; they must be appropriate to the situation.

3.0 CRATER BLASTING PRACTICES ELSEWHERE

Crater blasting practices are compared with like activities at the Homestake gold mine in South Dakota (Refs. 1, 2, 3, 4, and 5), the Carolusberg base metal mine of the O'Okiep Copper Company in South Africa (Refs. 14 and 15) and the CSA base metal mine at Cobar, New. South. Wales, Australia (Refs. 16 and 17). Crater retreat blasting is used to produce the total production of 108,840 tonnes (120,000 tons) per month from the Carolusberg mine and 60% of the total production of 181,400 tonnes (200,000 tons) per month from the Homestake mine. The CSA mine uses crater retreat blasting to mine all slot raises and produces 63,490 tonnes (70,000 tons) per month from longhole stoping with mass blasting.

3.1 Homestake Crater Blasting Practices

3.1.1 Background

In 1977 vertical retreat (VCR) mining was developed at the Homestake mine and by 1983 had become a fully integrated mining method contributing 60% to total underground production. VCR has proven successful for continuous ore blocks exceeding 45,350 tonnes (50,000 tons), with dips greater than 60 degrees. It provides a productivity average of 32 tonnes (35 tons) per man shift which is 92% greater than for cut and fill stoping.

Discontinuous ore bodies, small pods, or flat structures not amenable to bulk systems are mined with mechanized cut and fill (MCF) and have been able to achieve a 26 tonnes (29 tons) per man shift average productivity. Small, low grade, remnant ore bodies were not economically viable with large scale bulk techniques and could not support the development required for a MCF system. As a result, a modified version of VCR blasting with significantly diminished stope development was introduced in an effort to reduce capital

investment and expedite the production phase. This modified method is called small block vertical crater retreat (SB VCR).

3.1.2 Geology

The Homestake gold mine is located in the northern Black Hills of South Dakota at Lead. Having been active since 1876 it now extends from the surface to the 2,440 m (8,000 ft.) level underground. The host rocks are banded phyllites to schists and quartzite. The gold bearing Homestake rocks are schists to chlorite schists. The orebodies range in shape from tabular block to small clusters of pods.

3.1.3 General Mining

The majority of production comes from stopes exceeding 45,350 tonnes (50,000 tons), using panel mining (stope and pillar), VCR blasting techniques and cemented fill for regional ground control. The distance between the advancing face and the broken ore is invariably maintained at about 3.6 m (12 ft.) until orebreaking has been completed. A comparison between productivity and costs for the various mining methods practiced in the 6 year period from 1978-1983 (Ref. 2) is given in Table 4-4 below.

TABLE 4-4
HOMESTAKE PRODUCTIVITY AND COSTS (1978-1983)

MINING METHOD (tons/man shift)	PRODUCTIVITY per tonne (per ton)	NORMALIZED DIRECT MINING COST (per ton)
Vertical crater retreat	27.2 (30.0)	1.74
Blast hole	30.0 (33.0)	1.00
Shrinkage	16.3 (18.0)	2.23
Overhand cut and fill	13.4 (14.8)	2.53
Mechanized cut and fill	18.2* (20.1)	2.36*
	*(82, 83 only)	*(1983 only)

The productivity for VCR stoping is competitive with blasthole stoping and much superior to the other more manually intensive methods. The direct mining cost for VCR stoping is greater than that for blasthole, but less than the other methods.

The results from 5 trial stopes in the small block VCR mining program is described in Ref. 5. In summary it was found that productivity and cost/ton were comparable to those for mining large blocks with VCR. However, dilution was much higher in the small blocks as shown in Table 5-5 below.

TABLE 4-5
HOMESTAKE PRODUCTIVITY, COSTS & DILUTION (1988)

MINING METHOD	PRODUCTIVITY tonnes/manshift (tons/man shift)	NORMALIZED TOTAL MINING COSTS/TON	DILUTION (per cent)
Small block VCR	30.4 (33.5)	1.00	44
Large block VCR	31.7(35.0)	1.05	25
Mechanized cut & fill	26.5(29.3)	1.37	15
Overhand cut & fill	16.6 (18.4)	1.53	10

The comparisons show that VCR is a viable method in small ore blocks if extra dilution can be tolerated. Control of drillhole deviation is therefore critical to its successful application. Ground support requirements are much less than for either form of cut and fill stoping.

Studies at Homestake have compared the variation in total mining cost per ton with stope tonnage. Since total development costs for a large block or a small block VCR stope are similar, large blocks are more economically viable. Cost comparisons showed a 9% increase in total mining cost per ton as stope tonnage dropped from 117,910 to 68,025 tonnes (130,000 to 75,000 tons), and a 19% increase from 117,910 to 18,140 tonnes (130,000 to 20,000 tons).

3.1.4 Blasting Practices Small Block Mining

Dilution and Hole Deviation

Minimization of dilution is critical to the economics of a small block stope. In the hole (ITH) drill holes of 165 mm (6.5 in.) diameter have deviated up to 4.5 m (15 ft.) and when the small block is only 5 m (16 ft.) wide, this could result in 100% dilution. To correct this problem, holes are plotted with survey ties and fill in holes are drilled where necessary. To define the outside parameter of the small blocks, a dry drill sampler is most effective. If samples from the holes on the ends of the stope are barren, remaining holes in those rows are not drilled and barren holes are not blasted. Based on comprehensive small block data, this method averages 44% dilution, compared to 25% dilution for large block VCR, 15% for MCF, and 10% for OCF (Table 5). This high dilution factor makes control of hole deviation and dry sampling of outside holes critical to grade control.

Recovery statistics of 5 completed small block stopes are given in Table 6-6.

Drill Patterns and Blasting

Cleaner holes, break-out holes, and fan holes are used to drill out VCR stopes (Fig. 4-12). Cleaner holes are drilled parallel to and just inside the hanging wall of the ore body. These parallel holes produce a smooth

break along the hanging wall and minimize hanging wall caving and dilution. Break out holes are those that break out in the bottom sill.

Results indicated that rows with less than three break out holes presented difficulties. Specifically, the southeast end of 5 stope (Table 6-6) had several rows with only two or three break out holes. Problems encountered included plugged holes, fly rock on the topsill between blasts and sloughing of hole collars. The time required to correct the problems reduced mucking rates and increased man hours per ton. Rows with three or more break out holes were consistently blasted with success. Fan holes can be drilled into the footwall and provide a way of drilling and blasting ore that lies below and outside the sill. Fan holes are only used on the footwall side of the stope, where the possible gouging or overbreak rock will stay on the footwall and not introduce dilution.

Mass blasting, which has proven cost-effective in large VCR stopes also provides cost savings in the small block VCR stopes. To make a mass blast, one third of the stope volume is taken out by conventional vertical crater retreat blasting in 3 m (10 ft.) horizontal slices. Burden and spacing of holes are 3 m (10 ft.) respectively. Finally, through surface and in-the-hole delays, the remaining intact stope is blasted with a maximum of 225 kg (500 lbs.) of explosive per delay to eliminate any damage to surface or surrounding structures.

3.1.5 Conclusions

Compared with large block VCR mining, small stopes (under 45,000 kg (50,000 tons)) have the highest dilution factor. Therefore, controlling hole deviation and use of the dry drill sampler are critical to the method.

A minimum of three break out holes are recommended for successful blasting. Cleaner holes parallel to the hanging wall are mandatory to reduce dilution. Mass blasting of small block VCR stopes reduces costs and dilution.

Small block VCR mining appears to be an economically viable alternative to mechanized cut and fill mining for an isolated ore body if the stope has less than 34,100 tonnes (37,500 tons). Small block VCR is probably preferable to overhand cut and fill mining for stopes less than 32,000 tonnes (35,000 tons).

3.2 Carolusberg Crater Blasting Practices

3.2.1 Background

The stoping method at the Carolusberg mine is a classical example of the application of VCR blasting to control the sloughing of stope walls in deep mining. Stope dimensions are 18 m x 18 m (60 ft. x 60 ft.) in plan and are mined in 52 m (170 ft.) lifts from the bottom to the top of the various pod like orebodies. The stopes are mined in a checkerboard pattern and are filled immediately with cemented fill after completion of the draw.

The orebodies are comprised of a heavily jointed diorite intrusive and are bounded by strong, competent gneisses and granites. Current deep mining is between the 854 m (2,800 ft.) level and the 1,590 m (5,200 ft.) level, where the horizontal stresses range from 7300 psi to 13,500 psi. Previous longhole open stoping in the upper part of the mine was unsuccessfull, due to the caving of walls and backs under the influence of high horizontal stresses.

3.2.2 Drill Patterns and Blasting

Drill drives of 3.9 m x 4.2 m (13 ft. x 14 ft.) dimensions are developed at vertical intervals of 52 m (170 ft.). Fans of 165 mm (6.5 in.) holes are then drilled from these drives to ensure that charges can be placed on a nominal 3 m x 3 m (10 ft. x 10 ft.) grid. A high degree of drilling accuracy is expected and obtained. Once the drilling layout has been approved, a section is issued for each ring.

These sections detail the length and inclination of every hole in the ring, drawn to a scale of 1 in 500. Side pegs are installed by the Survey Department at each ring and these are used to ensure that the rig is lined up in the correct vertical plane. Once a stope has been completely drilled, each hole in the ring is surveyed and the data fed into a micro-computer which produces plots of the actual positions of all holes on plans at 2.5 m (8 ft.) vertical intervals. These plans are examined to determine whether any extra drilling is required to fill gaps that may have been produced from inaccurate drilling or hole deviation.

In addition, new ring sections are produced showing the actual position of each hole.

Charging is the most critical part of the stoping operation and is controlled directly by specialist shift bosses. After a blast, the remaining length of blasthole is measured as well as the distance to the top of the shrink pile. Both profiles are then marked on the relevant ring sections. The shift boss checks that sufficient clearance exists below the back to accept a blast and notifies the production crew to curtail or stop operations if the pile is at too low a level. He then selects the computer 2.5 m (8 ft.) hole plan that is nearest to the actual back elevation and, with the ring section, determines which holes should be blasted to give an approximate 3 m x 3 m (10 ft. x 10 ft.) grid, the timing of each hole and the required depth of burial.

The ideal stand-off distance for the charge of slurry explosive varies from stope to stope and during the life of an individual stope. It has been found that the horizontal stress tends to form the stope backs into a gently arch. Because of this, each blast is timed so that the first holes to fire are approximately at the highest point of the arch and the rest of the holes are timed in concentric rings around these first holes. Advances of up to 4 m (13 ft.) per blast are normal with burial depths of 1.9 m to 2.5 m (6 ft. to 8 ft.) and fragmentation is generally good. Powder efficiencies obtained average 0.34 kg/tonne (0.75 lbs/ton) for primary blasting and .09 kg/tonne (0.18 lbs/ton) for secondary blasting.

The first few vertical crater retreat blasts are taken together, with peripheral stripping blasts to attain the required stope dimension. Until the stope has reached the right size, the drawpoint is completely cleaned out after each blast to enable visual inspection of the stope back to ensure that breaking is satisfactory. The drawpoint brow raise assists in this initial blasting by providing an additional breaking face in the tight start

conditions. On occasions, extra drill holes have had to be drilled to assist in the initial break out. When breaking conditions are tight, the depth of burial is usually slightly less than when the stope is full size.

Stope backs are brought to about 12 m (39 ft.) from the footwall of the upper drill drive before holing procedures are initiated.

Initially, the practice was to sequentially blast three decks of spherical charges over the area of the stope to be holed, using the holes drilled from the drill drive. This was found to be unsatisfactory for a number of reasons, amongst which were loss of access in the event of a failed break through and excessive damage from concussion. The latest practice is to bring up a slot about 4 m (13 ft.) wide at one end of the stope and then retreat horizontally towards the other end using full column charges of ANFO.

3.2.3 Conclusions

Improved ground control in deep mining at the Carolusberg mine of the O'Okiep Copper Company has resulted from the application of VCR blasting techniques to permit large scale shrinkage stoping. This has been achieved by attention to detail in the practice of drilling and blasting and close supervision by specialist staff of the critical charging operation. Advances of 4 m (13 ft.) per blast are normal in the jointed rock, with the overall back forming a gentle arch in the near vertical, pod like orebodies.

The level of broken ore is not allowed to drop more than about 4 m (13 ft.) below the stope back, hence only 40% of the broken ore is mucked while blasting is in progress. The 12 m (40 ft.) crown or deck pillars are mined and holed using full column charges of ANFO. The initial practice of sequentially blasting three decks of spherical charges had to be abandoned because of excessive damage and problems due to loss of access.

Operators have had considerable success in the use of VCR blasting for the development of 3.6 m x 3.6 m (12 ft. x 12 ft.) raises up to 70 m (230 ft.) long in granite.

3.3 CSA Crater Blasting Practices

3.3.1 Background

The CSA copper/lead/zinc orebodies of Cobar Mines Pty. Ltd. are comparatively low grade. They occur in steeply dipping siltstones which have been cut by a wide, near parallel shear zone. The two lenses are located about 91 m (300 ft.) apart and lie along the edges of the host shear zone. Lens dimensions are erratic in plan, varying from 6 m to 21 m (20 ft. to 70 ft.) wide. Dip is parallel to cleavage at 75 - 80 degrees. The original cut and fill stoping method was replaced with longhole open stoping in 1978. Problems encountered in past stoping practices are discussed in more detail in Ref. 16.

Since 1983, a concerted effort has been made at the CSA mine to reduce dilution and improve fragmentation in longhole stoping. Some of the more significant changes made to stope design, drilling and blasting practices are listed below:

1. Reduction of blast hole diameter from 165 mm to 140 mm (6.4 in. to 5.5 in.).

2. Stopes designed, where practical, to enable parallel wall holes along the ore-waste contact.

3. Reduction of charge weight of explosive per delay number.

4. Use of diluted ANFO (30% ANFO) as the primary blasting agent.

5. Intensive supervision of charging operations.

6. Undercutting of stope sills.

7. Stope wall hole surveys.

8. Adoption of multi-row underhand benching as the stope blasting technique.

9. Change from all electric to non-electric firing system.

3.3.2 Drill Patterns and Blasting

Slot raises for long hole stoping are mined by VCR blasting using 165 mm (6.4 in.) diameter blastholes over a drill level interval of 38 m (125 ft.). The same rod strings (115 mm (4.5 in.) outside diameter) are used to drill 140 mm (5.5 in.) diameter holes for bench blasting on a 3 m (10 ft.) burden with a maximum toe spacing of 3.6 m (12 ft.). Previous experience with 114 mm (4.4 in.) blast holes over the same drilling interval yielded excessive deviation. Collar holes are surveyed and camera surveys are made of all wall contact holes. Experience has shown that set up errors are a problem and that there is a need to increase set up accuracy.

Slot raises are located in the centre of the stope, away from the walls to reduce the damage associated with VCR blasting of 165 mm (6.5 in.) diameter holes, fully loaded with ANFO. Blasting procedure was to advance the slot raise to within 5 m (16 ft.) of breakthrough, then extend the slot raise to ore limits before commencing bench blasting in the main stope. However large blocks were torn away near the base of the stope using this procedure (Figure 4-13). The blasting sequence was modified to advance the stope in three phases of underhand bench blasting to achieve the shape shown for the final blast in Figure 4-14. A major contributor to improved fragmentation resulting from the modified blasting sequence was the experimentation which permitted the use of diluted ANFO (30% ANFO) for slot extension and all production blasting, except initial breakout of the undercut. Full strength ANFO is still used to advance the slot raise in 5.5 m (18 ft.) increments (165 mm (6.4 in.) diameter blastholes) and for some break out holes in the bottom lift (140 mm (5.5 in.) diameter blastholes).

Intensive supervision of charging operations has been implemented using full time employees. This was necessary to ensure control of the complicated blasting process and the implementation of correct procedures.

3.3.3 Conclusions

Drop raising combined with underhand benching has been used successfully at the CSA mine at Cobar to reduce dilution. Current stoping practices in the sheared ground are an evolution from cut and fill mining through conventional longhole stoping.

A major contributor to the final result of reduced dilution and improved fragmentation is the use of diluted ANFO for slot extension and for production blasting. It has also been shown that charge weights resulting from full column loading of 18 m x 14 cm (60 ft. x 5.5 in.) diameter blastholes do not result in excessive overbreak in weak ground, avoiding the need for deck charging.

Drop raises can be successfully advanced in weak ground using VCR blasting with full strength ANFO in 165 mm (6.4 in.) holes.

Intensive supervision of charging operations is needed when blasting procedures must be followed closely.

3.4 Summary and Conclusions

The common threads in the analysis of stoping and blasting practice at the three mines discussed above can be summarized as follows:

1. Stoping procedures and blasting practices must be tailored to accommodate the ground conditions, geometry of the orebodies and the stress environment.

2. Pure VCR mining techniques seem more applicable to large scale orebodies in competent rock.

3. Hybrid blasting techniques seem more applicable to steeply dipping and narrower orebodies in medium to weak rock.

4. Specialist supervision is essential to optimize VCR blasting techniques.

5. Dilution is a problem when mining small blocks in medium to weak ground using long blastholes (plus 30 m (100 ft.) long). Basically all blastholes should be drilled within the orebody.

6. Hole deviation seems to be endemic to longhole blasting; results from VCR blasting are particularly sensitive to hole deviation.

7. The wall support offered by shrink stoping using VCR blasting is a major incentive to test the method in large pod like orebodies of the type mined at Carolusberg.

8. The VCR blasting technique has considerable application for the development of slot raises.

9. Explosives must be matched to the ground conditions, and the blasting techniques used must satisfy this criteria.

4.0 BLASTING RESULTS - 1987 STUDY AT THOMPSON

4.1 General

There is a perception amongst operators and engineers that blasting results in terms of explosive performance, fragmentation and damage can only be improved through close attention to key aspects of the art which include:

- Improved blasthole drilling accuracy.
- Improved initiation systems.
- An understanding of the sphere of influence of a blast.
- Measurement and understanding of the influence of in-situ rock properties and geological structure on blasts.

Dilution can be reduced by improving blasthole drilling accuracy and spacing sublevel intervals to conform more closely to the geometry of orebodies, particularly narrow orebodies. Equipment and techniques are available to decrease "set up" errors resulting from poor alignment of the drill rig and to monitor hole deviation to achieve design objectives. At Thompson, blasthole deviation is checked by surveying break through locations and plotting the results in plan. Deviation is not measured in non-break through holes, but the depth at which ore is intersected is measured and the portion of the hole in waste is plugged.

The results of a study carried out in 1984 to assess hole deviation at three mines in the Sudbury basin is given in Table 4-7. An assessment of these results presented in Table 4-7 indicate that average deviation of long blastholes can approach 5% in dipping strata. Deviation of this magnitude would be excessive in the narrower orebodies at Thompson, and it is clear that layouts should be modified to maximize the number of break through holes into the undercut. A survey is needed to sample the current situation as a basis for assessing whether additional controls and/or design changes are necessary.

During the last 10 years, blasting practices have been upgraded at a large number of mines using blast vibration monitoring techniques (Hammett et al, 1988). Explosive charges that cause high vibration levels can be located. Also, misfires can be detected and the actual sequence of detonation can be compared with the designed sequence. A limited blast vibration monitoring program was undertaken at the Thompson mine in 1987 (Ref. 6) and some of the important results are discussed below.

4.2 Blast Monitoring Program

A total of thirteen blasts were monitored at Thompson during 1987, including three deck blasts (Ref. 6). Test locations are shown in Figure 4-4 and are listed below:

- 1400L - 3 Shear (deck blast only).
- 2200L - 238 Stope (one blast).
- 2200L - 32 Stope (four blasts plus deck blast).
- 1803L - 331 Stope (four blasts plus deck blast).
- 1603L - 340 Stope (three blasts).

The stopes monitored differed in varying degrees with respect to rock characteristics and blasting parameters. However, misfires were identified in all of the thirteen blasts monitored. In some instances, more than 30% of the explosive charge did not contribute to rock breakage. Misfires were largely caused by "proximity" effects, whereby an explosive charge breaks more than its planned burden and interferes with an adjacent, later delayed charge. This result can be caused by overcharging, blasthole deviation beyond design limits, ground conditions or a combination of all of these factors. In summary, the limited program of blast vibration monitoring highlighted the need for further investigation of the following parameters.

- Drill hole pattern.
- Hole diameter and hole deviation.
- Explosive type.
- Stemming.

4.3 Blast Monitoring Techniques

Vibrations from geophones positioned adjacent to a blast were recorded with a tape recorder and then digitized using an oscilloscope (current technology utilizes a PC-based digitizing, recording and analysis system). The first step in analysis was to assign discrete vibration impulses to a particular deck within the blast. However, the degree of confidence attached to each of the interpreted detonation events depends on the spatial distribution of the geophones relative to the explosive charges. Also, the complexity and scale of blasting influences the relative ease of interpretation. An isometric representation of the placement of geophones for monitoring the blast vibrations in individual stopes is shown in Figure 4-15. The provision of only two arrays did not permit unambiguous interpretation of the source of all vibration impulses, but the priority was to overview blasting practices in several VBM blocks in a limited campaign. In interpreting results, it was assumed that delay scatter was small enough to avoid sequence reversals. A simple wirebreak mechanism attached to the starter detonator

provided a zero time reference to calculate the actual time of detonation. Arrival times of vibration impulses at individual geophones were corrected by calculating and subtracting travel time.

4.4 Case Study

The vibration monitoring program carried out in 1803-331 VBM Stope - 4 Block is summarized as an example of the field procedures used and the interpretation of results. Details of the stope geometry and blasts monitored in 4 Block are given in Figures 4-7a to 4-7g inclusive. Orebody statistics are summarized below:

- Height from undercut to overcut, 61 m (200 ft.) (Fig. 4-7b).
- Ore width, variable from 10ft at the undercut level to a maximum of 18 m (60 ft.) at the overcut level (Fig. 4-7a).
- Dip of the hangingwall varied from 75^{o} at the undercut level to 55^{o} at the overcut level.
- Strike length as defined by the deck blast, 14 m (45 ft.).
- Ore extraction by longitudinal retreat.

The blasts monitored are shown in section on Fig. 4-7b and in plan on Figures 4-7c - 4-7f inclusive. Figure 4-7g is a record of the blast vibrations recorded in the deck blast.

Significant blast design parameters are presented below:

•	Hole diameter	- 165 mm (6.5 in.)
•	Blast-row spacing	- 1.5 m (5 ft.)
•	Drill pattern	- Blastholes drilled as inverted fans, collar spacing 0.6 m (2 ft.) and toe spacings 6 m (20 ft.) to 8 m (25 ft.)
•	Number of holes drilled	- 41 (Rows C - L inclusive)
•	Stand off distance	- 1.8 m (6 ft.)
•	Average breakage per blast	- 2 m (6.5 ft.)
•	Explosive type	- Mainly Aquamex.

VBM blasts 8, 9, 12 and 16 (Fig. 4-7b) plus the deck blast were monitored. The locations of the four monitoring stations (B4 - B7 inclusive) are shown in Fig. 4-7a. No information was available on drilling accuracy. The breakage diameter attributed to each blasthole was assumed at 3 m (10 ft.).

A major design deficiency of the blast layout was the spatial arrangement of the charges. Lateral distribution was far from uniform and posed problems in terms of muck size and potential charge interference.

Blast vibration records gave the following results (Table 4-8).

- Blast No. 8 - 2 of the 13 charges failed to detonate.

- Blast No. 9 - 3 of the 15 charges failed to detonate and four holes were plugged after the blast. (Table 9).

- Blast No. 12 - 1 of the 19 charges failed to detonate, but an additional three charges detonated at very low efficiency (Table 4-10), indicating damaged or dislodged charges. Figure 4-7d shows the nominal blast geometry and it can be seen that there is much more overlap of charges compared with blast No. 9 shown in Figure 4-7c.

- Blast No. 16 - 2 of the 25 charges failed to detonate (Table 4-11), which is remarkable, considering the bunched nature of the blast holes shown in Figure 4-7e.

- Deck Blast - The thickness of the top sill fired in one blast was about 9.1 m (30 ft.) towards the centre and 18 m (60 ft.) towards the adjacent filled stope (Rows D and C). Twenty six holes were charged from rows C to L inclusive (Fig. 4-7a) giving 81 explosive deck charges of either 25 kg (55 lbs) or 37 kg (82.5 lbs) weight. The plan view of the nominal charge distribution is given in Figure 4-7f and the radial component of the blast vibration record is presented in Figure 4-7g. Charging and sequencing details are presented in Table 4-12.

The most significant feature of the blast is the almost total malfunction of charges in Rows F through to C. Development of the slot went as planned, although it is likely that some decked charges were detonated sympathetically e.g., Holes H2 and L2. In many other instances, a significant proportion of the decked charges in any one hole failed to detonate; a good example is Hole J4 containing three decks each at 725 ms. One charge detonated with full energy, one charge partially detonated (i.e. at low efficiency) one failed to detonate.

It is suggested that the rate of misfires detected in Rows I to L is probably typical of all deck blasts. The much higher incidence of misfires toward the previously mined and filled stope (Block #3) is undoubtedly due to ground dislocation. This event would be directly attributable to the development of the slot and subsequent removal of support.

4.5 Discussion and Summary

The case study outlined above illustrates the application of blast vibration monitoring, but it is important to review the results in the light of the overall study of the VCR blasting technique. The high incidence of misfires recorded and summarized in Table 4-8 is indicative of the need to revise blast design criteria and reduce explosive consumption to prevent loss of control of the blasting process. Improved blasting provides the potential for reduced powder factor and improved fragmentation.

The first objective in design is to provide more uniform explosive distribution and sufficient separation between charges to ensure that the explosive placed is fully utilized. Whilst crater testing defines the optimum depth of burial (burden) for the given operating conditions, it does not provide any useful measure of the powder factor necessary for rock breakage. Determination of blasthole spacing in production blasting is a more subjective process and as a first estimate, spacing up to twice the burden may be practicable. However, blasthole spacing (and powder factor) must reflect the rock mass, which is weakened by persistent structure at Thompson.

Variable blasthole arrangements at successive stope elevations resulting from "fanned" (non-parallel) hole drilling complicates the blast design. Hence, the limiting range of charge-to-charge spacing needed to achieve complete detonation of all charges has to be determined; blast vibration monitoring will assist in this regard.

In VCR blasting, breakage of rock is most intense in the (crater) burden volume as defined by free-face geometry, and the displacement action of the expanding explosive gasses. However, this is not representative of the total breakage mechanism. Breakage beyond the placement of the charge, whilst a bonus in terms of reduced blast cycles is largely structurally controlled. Structure is a possible cause of oversize and by extension, overbreak into bounding schists should be expected from holes located adjacent to ore limits.

A review of VCR blasting inevitably invites the comparison with slash or bench blasting. Fundamental differences in the rock breakage mechanism exist between the two blasting methods. Observations of VCR blasting at Thompson show the relation between fragmentation, powder factor and overbreak. VCR blasting compounds the problem of damage and overbreak by introducing an added degree of structural instability. One inference from this assessment is that VCR blasting at Thompson does not lend itself to large scale blasting using decked charges.

Blast vibration monitoring highlighted the potential for damage to schists bounding VBM stopes. Both ANFO and Aquamex explosives in 165 mm (6.5 inch) diameter holes generate sufficient strain energy to cause damage as evidenced by large scale ground failure during deck-blasting. An experimental blasting program to evaluate and compare different explosive types, including diluted ANFO seems appropriate.

Blasthole drilling accuracy, although not part of the blast vibration monitoring program had a major effect on fragmentation and was considered to be a major cause of dilution. Breakthrough survey data were available for only two of the five blocks monitored, but implications were significant.

In one block surveyed, drilling accuracy over the total hole depth of about 43 m (140 ft.) was reasonable; of 7 breakthrough holes surveyed, deviation exceeded 1.5 m (5 ft.) in only two instances. However, the maximum recorded deviation of 2.1 m (7 ft.) equates to 5% deviation as recorded at one mine in the Sudbury basin which had dipping strata (Table 4-7). The evidence to date is that a deviation exceeding 1.2 m (4 ft.) is not tolerable for a nominal toe spacing of 3 m (10 ft.).

The second block surveyed highlighted the problems incurred by inaccurate drilling and by failure to use surveyed breakthrough data in the blast design. In six instances, adjacent holes less than 1.2 m (4 ft.) apart were charged. 33 per cent of the charges misfired.

In summary, it is very important to survey breakthrough holes and use the information in blast design.

4.6 Conclusions

1. Blast vibration monitoring identified misfires in all VBM blasts studied (thirteen in all). In some instances more than 30 per cent of the explosive charge did not contribute to rock breakage.

2. Misfires were largely attributable to "proximity effects", whereby an explosive charge breaks more that its assigned burden and interferes with an adjacent, later-delayed charge. This mechanism may be the result of over charging, in-situ structure and/or drillhole deviation.

3. Blast diagnosis indicated the need to redefine blast design criteria and evaluate rock mass characterization in terms of the rock breakage process. The net result of the current lack is excessive explosive consumption, misfires and the creation of oversize muck.

4. Deck blasts represent a hybrid design, incorporating elements of both crater and slash blasting; blast execution is further complicated by complex decking and sequencing procedures, coupled with inherently unstable ground condition.

5. Examination of muck revealed fragmentation to be predominantly fine in size, although a number of large boulders were generally present. Bi-modal muck size distribution is characteristic of overcharging where the fine product comes from the true crater volume while the large fragments are generated by overbreak and fall-off.

6. Current powder factors of the order 0.4 to 0.75 kg/tonne (0.8 to 1.5 lb/ton) are considered to be excessive for the rock mass condition. The extent to which high powder factor values are attributable to VCR blasting or simply to overcharging is difficult to apportion. However, there is scope to reduce explosive usage because of the high rate of misfires recorded.

7. Optimum explosive distribution for crater breakage mechanisms requires parallel holes drilled on an equivalent spatial basis such as a staggered, equilateral pattern. However, relatively narrow overcut drill drives necessitate "fanned" drill rings, resulting in a variable arrangement of holes at successive blast elevations.

8 Whilst depth of burial (burden) of the charge is well defined by crater tests, spacing between holes needs to be determined experimentally. As a first estimate, spacing of the order of twice the depth of burial may be a practical objective to work toward.

9. The generally accepted need for high charge concentration in crater blasting (large hole diameter and high bulk strength explosives) presents a contradiction in terms of the efficiency of explosive action. Excessive charge concentration increases the potential for overbreak. The effect of charge concentration on breakage (rate of advance) and fragmentation should be carefully assessed.

10. VCR blasting is characterized by relatively small scale blasting (in terms of blasthole stoping) and consequently a large number of blast cycles. Variable breakage from crater blasting effectively prevents charge decking and this restricts blast size.

11. Blast vibration monitoring provides the most effective means of assessing blast design modification and quantifying blast performance. The essential techniques for blast optimization are rock mass structural characterization, blast vibration monitoring and fragmentation assessment.

5.0 DESIGN GUIDELINES FOR DRILLING AND BLASTING

Currently, nearly all of the ore in VBM stopes is broken using conventional VCR blasting techniques. Supplemental multi-deck blasting procedures are used to expand blocks laterally by slashing and to recover crowns. It has been shown that the cost advantages and higher mining efficiencies (compared with mechanized cut and fill stoping), have resulted from the application of VCR blasting techniques. Howevever, these advantages have been offset in part by very heavy dilution in narrow, sinuous orebodies and reduced ore recovery compared with traditional mining and/or rock breakage approaches. As noted previously, this is a conscious trade off, but it seems that the original concept of VBM as defined in the introduction is still appropriate, particularly the observation relating to the application of drilling and blasting practices. It is beyond the scope of this report to detail alternative rock breakage approaches that may be applicable, but it is suggested that full column bench blasting has a future role in stoping operations. The remaining discussion is focused mainly on guidelines for current practices, considering rock breakage, fragmentation and damage aspects.

5.1 Blasthole Drilling Practice

1. Blastholes should be drilled parallel to the hangingwall. Any fan holes drilled should intersect the footwall only.

2. Three blastholes per ring should break through into the undercut, with a minimum of two breakthroughs.

3. Break through holes should be surveyed and tied back to their collars; the information should be used to revise the blast design as appropriate, including the drilling of additional blastholes as needed.

4. The portion of blastholes drilled beyond design wall limits should be plugged 1 - 1.2 m (3 - 4 ft.) inside the design wall limits.

5. Considering damage, 114 mm (4.5 in.) blastholes should have preference over 165 mm (6.5 in.) blastholes.

5.2 Blasting Practice

1. The powder factor should be reduced to match rock condition. Aim for a powder factor of less than 0.22 kg/tonne (0.5 lb/ton) for VCR blasting operations in narrow orebodies, based on the use of Aquamex or equivalent explosive.

2. Eliminate current deck blasting practices to reduce damage. Use full column charging with diluted ANFO to bench the crown after establishing a slot. Reduce hole diameter by artificial means to further reduce damage if necessary.

3. Test the method used at the Homestake mine to extract about one third of a stope block by conventional VCR blasting and the remainder in a continuous full column bench blast using diluted ANFO.

4. Standardize drilling and blasting practices throughout the mine using specialist supervision as practiced at the Cobar mine.

5. Reduce explosive consumption by determining the limiting range of charge to charge spacing needed to avoid charge malfunction or dislodgement.

6. Monitor blasting practices regularly to maintain standards and test new procedures.

5.3 Future Work

5.3.1 Hole Deviation

Hole deviation seems to be endemic to longhole blasting in dipping orebodies. Results from VCR blasting are particularly sensitive to deviation. At the mines surveyed, breakthrough holes are tied back to their collars and at the Cobar mine, camera surveys are made of all wall contact holes. Equipment and techniques are available to decrease "set up" errors resulting from poor alignment of the drill rig and to monitor hole deviation to achieve design objectives. The results of the limited investigation reported in 1987 (Ref. 6) indicates that there could be a problem at Thompson with blast hole drilling inaccuracy. A project should be initiated to determine the present position so that current operating procedures can be revised as necessary. The Cobar mine uses minimum size bits to stiffen their drill strings and reduce hole deviation; this procedure can be tested early in operations.

5.3.2 Blasting Results

Variable blasthole arrangements at successive stope elevations resulting from fanned (ring) drilling complicates blast design. The first revision of an initial design follows the receipt of survey results at completion of the drilling program. Succeeding revisions are usually required at each lift, based on the results of hole depthing and an assessment of the holes available for charging. Considerable thought needs to be given to how a

go, no-go situation should be handled in view of the pressures of production. In this regard, it is clear that schists bounding the stope have well developed structure, sufficient to cause continuous dilution once the hangingwall is breached.

VCR blasting is characterized by relatively small scale blasting and consequently a large number of blast cycles. Variable breakage from crater blasting effectively prevents charge decking, hence it seems important that a block should be mined to test the combined VCR, mass blast configuration as practiced at the Homestake mine. The blasting procedure practiced at the Cobar mine should also be evaluated for potential application.

The measuring and charging procedures for VCR blasting, although relatively straight forward, must be closely followed if satisfactory results are to be obtained. At the Homestake mine, close attention is paid to maintaining a relatively flat back profile after each blast and in this regard, a shrink pile of broken rock helps maintain the stability of the stope.

6.0 REFERENCES

1. Mitchel, S.T., (1980). Vertical Crater Retreat Stoping Proves Successful at Homestake Mine. Mining Engineering, Vol. 32, No. 11, Nov., pp 1581-1586

2. Orr, S.A., (1983). Refinement of Vertical Crater Retreat Stoping at the Homestake Mine. Paper presented at Southwest Section of AIME, Tucson, Arizona, Dec.

3. Andrews, R., (1984). Mass Blasting at the Homestake Mine. Homestake Mining Coy., Lead, South Dakota.

4. Haptonstall, J.C., (1984). Success and Failures of VCR Mining at Homestake. Homestake Mining Coy., Lead, South Dakota.

5. Muir, J., (1988). Small Block VCR Mining at Homestake. Mining Engineering, Sept., pp 889-892.

6. Golder Associates, (1987). Investigation of Blasting Practice in T 1 and T 3 Mines. Consulting Report to INCO Ltd., Thompson, Manitoba, Sept.

7. Sarin, D.K., (1981). Vertical Block Mining at INCO'S Thompson Mine. 5th. Annual Underground Operator's Conference, Sudbury, Ontario, Feb.

8. Stewart, G. and Maclean, R. (1988). New Developments in the Mining of Underground Pillars at Thompson Mine. C.I.M. Bulletin, Oct., pp 54-58.

9. Bryant, P.E. (1986+). Vertical Crater Retreat Stoping at O'OKiep Copper Company. O'OKiep Copper Company Ltd.

10. Anon (1985). The Rock Mechanics Aspect of the Carolusberg Deep Ore Mine. O'OKiep Copper Company Ltd.

11. Munro, S.J. and Jackson, G.S. (1985). Longhole Open Stoping Practice at the CSA Mine Cobar. The Aus.I.M.M. Kalgoorlie Branch, Underground Operator's Conference, Oct., pp 89-96.

12. Price, I.L. and Haskard, R.W. (1983). Longhole Raising at the CSA Mine, Cobar. The Aus.I.M.M. Conference, Broken Hill, July.

13. Hammett, R.D., LeJuge, G.E. and Forsyth, W.W. (1988). Diagnosis and Optimization of Blasting Problems and Procedures Through Blast Monitoring, 90th Annual General Meeting of CIM at Edmonton, April.

14. McKenzie, C.K., Dawes, J.J. and Chitombo, G.P. (1986). The Role of Vibration Monitoring in Blast Assessment. Mining Latin America, IMM, Nov., pp 231-240.

7.0 BIBLIOGRAPHY

1. Ashby, I.R. and Bradford, N.L. (1985). Use of Isonal in Production Blasting at Isa Mine. The Aus. I.M.M. Kalgoorlie Branch Proceedings of the Underground Operators Conference, Oct., pp 157-166.

2. Lang, L.C., Roach, R.J. and Osoka, M.N. (1977). Vertical Crater Retreat an Important New Mining Method. C.I.M. Bulletin, Sept., pp 69-76.

3. Lang, L.C. (1978). Cratering Theory Evolves into New Underground Mining Techniques. The Aus. I.M.M. Melbourne Branch, Rock Breaking Symposium, Nov., pp 115-124.

4. Edwards, D.R. (1983). Vertical Crater Retreat Pillar Mining - A Case Study. The Aus.I.M.M. Conference, Broken Hill, July, pp 217-223.

5. King, B.M., Just, g.D. and McKenzie, C.K. (1988). Improved Evaluation Concepts in Blast Design. Trans. Instn. Min. Metall. (Sect. 11:Min. Industry), 97, Oct., pp A173-A181).

6. Just, G.D. (1978). Evaluation of Alternative Blast Designs. Transactions I.M.M., January.

7. Cameron, A. and Scott, A. (1988). The Field Performance of Explosives. Explosives in Mining Workshop, Aus. I.M.M., Melbourne, November.

8. Mathews, K.E. and McKenzie, C.K. (1985). Trends in Blasting Research. 87th Annual General Meeting of CIM at Vancouver, April.

9. Bywater, S. and Morland, R.L. (1987). Recent Developments in Cable Support and Blasting Research at Mount Isa Mines Ltd. Presentation at AMIRA Technical Meeting on Advances in Blasting and Ground Support Held at Sydney, N.S. Wales, October.

10. Mathews, K.E. (1987). An Overview of the Interdependence of Blasting and Ground Support in Mining. Presentation at AMIRA Technical Meeting on Advances in Blasting and Ground Support Held at Sydney, N.S. Wales, October.

TABLE 4–3

1403 – 301 VBM STOPE MINING STATISTICS

	BLOCK 1	BLOCK 11	BLOCK 2	BLOCK 4	BLOCK 3	BLOCK 6	BLOCK 5	BLOCK 8	BLOCK 7	BLOCK 10	BLOCK 9	BLOCK 13	BLOCK 12	BLOCK 14
AVE. BREAK (FT.) PER CRATER BLST.	4.6	N/A	4.1	5.3	5.7	5.3	6.6	8.3	6.0	8.0	7.6	4.6	7.9	7.2
AVE. DESIGN WIDTH (FT.)	8	13.5	12.5	11.1	10.9	12.7	15.9	18.7	16.5	13	13.4	15.6	26	11.1
AVG. OVERBREACK (FT.)	6.5	5.2	-1.2	0.4	1.93	-1.9	4	6	5.3	8.8	-3.7	6.9	N/A	4.0
AVG. TOTAL BLOCK WIDTH (FT.)	14.5	18.7	11.3	11.5	14.8	12.1	17.8	16.8	20.1	19	18.7	24.4	22.3	18
AVE. ACTUAL ORE WIDTH (FT.)	7.6	13	8.3	5.8	7.9	9.1	11.5	13.9	11.5	10.1	14.2	23.3	10.9	-
MINTAB AVE, STRESS (MPa)	-	-	26	30	19	37	23	40	26	42	30	34	34	46
STRIKE AVE. STRESS (MPa)	88	95	52	48	48	50	50	48	64	51	59	44	40	55
BURDEN:SPACING (FT.:FT.)	6.0:4.0	6.0:5.0	5.5:4.0	4.8:4.0	5.2:4.0	6.0:5.0	5.4:4.0	5.0:5.0	5.4:5.0	5.0:5.0	5.3:5.0	6.0:6.0	7.5:6.0	6.0:6.0
AVE. AREA PER HOLE (SQ. FT.) AT BREAK THROUGH	14.3	19.5	12.3	12.7	13.5	14.5	14.4	16.4	24.2	19.7	19.2	25.7	31.7	29.8
POWD. FACTOR (LBS. / TON)	1.13	N/A	1.15	0.93	1.0	0.61	0.56	0.85	0.5	0.81	0.95	0.99	0.51	1.19
CHARGE WT. (LBS.) PER DELAY: AVE	47.4	N/A	22.9	27.6	22.4	21.7	26.5	31.5	29.6	28.3	29.6	N/A	24.3	27.0
MAX	168	N/A	88.9	56.8	28.8	272	72.5	105	86.5	51.4	65.1	N/A	70.7	216
STANDOFF (FT.)	3.0	N/A	3.4	4.3	4.7	4.0	4.1	4.5	4.7	4.1	3.6	N/A	3.8	3.0
BLOCK WIDTH	7.5	14.6	6.1	8	9.3	9	12.1	16.6	15.6	11.2	10.4	14.0	17.3	13.0

TABLE 4-6

HOMESTAKE RECOVERY STATISTICS - SMALL BLOCK VCR

	ESTIMATE			ACTUAL					
LOCATION	TONS	OZ.	GRADE	TONS	GRADE	OZ.	COST/TON	COST/OZ.	TPMS
7L-1850	28,500	5,515	.194 $4.00	42,473	.122 $2.52	5,182	$40.38	$332	37.4
9L-2300	30,000	4,572	.152 $3.15	54,003	.127 $2.63	6,858	$45.92	$363	29.1
9L-4700	12,600	3,536	.281 $5.80	14,678	.156 $3.21	2,290	$46.09	$296	23.0
17L-4700	43,200	8,639	.172 $3.55	54,081	.132 $2.72	7,117	$44.29	$335	29.7
17L-4700	42,900	9,755	.277 $4.70	65,128	.140 $2.89	9,118	$41.59	$298	45.6
TOTALS	157,200	32,017		230,363		30,565			
AVERAGES			.204 $4.21		.133 $2.74		$43,330	$326	$33.5

NOTE: TONS PRODUCED = 44% DILUTION FOR 73,163 TONS @.0198 GRADE.
OZ. PRODUCED WERE 5% UNDER ESTIMATE.

2-T6-1605

TABLE 4–7

DRILL DEVIATION STUDIES

NUMBER OF HOLES SURVEYED	MINE	LAYOUT DIP	AVERAGE ERROR DUE TO MACHINE POSITION (FT)	AVERAGE ERROR DUE TO DIP ALIGNMENT (FT)	AVERAGE ERROR DUE TO AZIMUTH ALIGNMENT (FT)	AVERAGE NET SETUP ERROR (FT)	AVERAGE ERROR DUE TO DRILL DEVIATION (FT)	AVERAGE ACTUAL TOE TO LAYOUT TOE (FT)	AVERAGE HOLE LENGTH (FT)	AVERAGE GYROPROBE ACCURACY (FT)	MACHINE
20	A	-90 TO-83	1.85	2.05	1.46	3.44	3.17	3.46	200.3	1.86	GO - 60
16	B	-87	0.75	1.98	1.66	3.09	3.04	3.98	185.74	2.64	JOY
14	C	-58 TO -51	1.47	7.17	9.76	13.12	9.09	10.71	198.4	-	GO - 60

NOTE: (1) ROD DIAMETER: 5–1/4"

(2) BIT SIZE: 6–1/2"

(3) GYROSCOPIC BOREHORLE SURVEY DEVICE CALIBRATED AGAINST SURVEY PICKUPS OF BREAK THROUGH HOLES.

TABLE 4-8

VBM STOPE BLAST DIAGNOSIS

VBM STOPE / BLAST DESIGN PARAMETERS	BLAST / NUMBER OF CHARGES / PREDOMINANT EXPLOSIVE TYPE	COMMENTS
1803 - 331 H/W Block #4 6 1/2" hole dia. 5' row spacing 20' toe spacing	#8. 13 decks; MAGNAFRAC #9. 15 decks; MAGNAFRAC #12. 19 decks; AMEX #16. 25 decks; AQUAMEX Deck 81 decks; AQUAMEX	2 misfired charges 2 misfired charges 4 misfired charge 3 misfired charges A large number of misfires partly attributable to fall-off. Ground froze in stope.
1603 - 340 Stope 4 1/2" hole dia. 5' roe spacing 5' toe spacing	#4. 12 decks; AQUAMEX #5. 20 decks; AQUAMEX #8. 47 decks; AQUAMEX	1 misfired charge 7 misfired charges >15 misfired charges
2201-32 Stope 6 1/2" hole dia. 10' row spacing 10' toe spacing	#12.19 decks; AMEX #15.16 decks; AMEX Deck 70 decks; AQUAMEX	2 misfired charges 2 misfired charges >20% misfire rate
2801 - 238 Stope 6 1/2" hole dia.	#1 24 decks; AQUAMEX	7 misfired charges
3 shear 6 1/2" hole dia.	Decks 91 decks; AMEX	>15% misfire rate

TABLE 4–9

BLAST DETAILS AND DIAGNOSIS, BLAST #9

HOLE	CHARGE DEPTH (FT)	EXPLOSIVE TYPE	CHARGE WEIGHT (LB)	NOMINAL DELAY (MS)	ACTUAL DETONATION (MS)	DELAY ERROR (%)	BREAKAGE (FT)	BLAST INTERPRETATION COMMENTS
I2	81	Magnafrac	44	190	174.8	-8.0	plugged	
H2	81	Magnafrac	44	280	269.0	-3.9	6	
K2	80	Magnafrac	44	340	374.4	+10.1	<10	
I1	84	Magnafrac	44	490	515.0	+5.1	7	
H3	81	Magnafrac	44	490			6	failed to detonate
J1	85	Magnafrac	44	570	585.4	+2.7	7	
J3	81	Magnafrac	44	570	609.5	+6.9	9	
G1	85	Magnafrac	44	650	699.2	+7.6	6	
F2	91	Aquamex	82.5	725			plugged	failed to detonate
K1	81	Magnafrac	44	725	776.7	+7.0	<10(P)	
L1	93	Aquamex	82.5	800	819.0	+2.4	8	
K3	81	Magnafrac	44	800	827.3	+3.4	8	
L2	76	Aquamex	55	875	906.0	+3.5	4	
I3	94	Magnafrac	66	950	1042.6	+9.0	plugged	wet hole
I3	84	Magnafrac	66	950				wet hole; failed to detonate

2/T9/1605

TABLE 4–10

BLAST DETAILS AND DIAGNOSIS, BLAST #12

HOLE	CHARGE DEPTH (FT)	EXPLOSIVE TYPE	CHARGE WEIGHT (LB)	NOMINAL DELAY (MS)	ACTUAL DETONATION (MS)	DELAY ERROR (%)	BREAKAGE (FT)	BLAST INTERPRETATION COMMENTS
I2	59	Amex	55	190	209.6	+10.3	plugged	Booster only
J2	59	Amex	55	230	259.1	+12.7	5	
H2	63	Amex	55	280	333.9	+19.3	plugged	
K2	70	Amex	55	340	381.7	+12.3	13	
I2	63	Amex	55	490	502.7	+2.6	plugged	Charge dislodged
H3	59	Amex	55	570	607.3	+6.5	5	
G1	67	Amex	55	650	700.7	+7.8	>4	
K1	70	Amex	55	725	743.7	+2.6	13	
L1	77	Amex	55	800	842.4	+5.3	0	
L2	67	Amex	55	875	917.7	+4.9	7	
J3	63	Amex	55	950	1023.7	+7.8	>4	
F2	76	Amex	55	1025	1072.3	+4.6	>14	Charge dislodged
H1	84	Aquamex	82.5	1125	1175.3	+4.5	>20	
H1	74	Aquamex	55	1125	1194.6	+6.2	>20	Charge dislodged
J1	62	Amex	55	1225	1317.1	+7.5	5	
K3	69	Amex	55	1400	1504.6	+7.5	16	
I3	67	Amex	55	1675	1794.2	+7.1	7	
L3	84	Amex	82	1950	1995.0	+2.3	31	
L3	74	Amex	82	1950	1999.2	+2.5	31	

TABLE 4–11

BLAST DETAILS AND DIAGNOSIS, BLAST #16

HOLE	CHARGE DEPTH (FT)	EXPLOSIVE TYPE	CHARGE WEIGHT (LB)	NOMINAL DELAY (MS)	ACTUAL DETONATION (MS)	DELAY ERROR (%)	BREAKAGE (FT)	BLAST INTERPRETATION COMMENTS
I2	30	Aquamex	55	190	215.4	+13.4	plugged	
J3	31	Aquamex	55	230	243.1	+5.7	11	
J2	30	Aquamex	55	280	289.7	+3.4	>3	
K2	30	Aquamex	55	340	378.0	+11.2	9	
H3	30	Aquamex	55	410	414.5	+1.1	plugged	
H2	32	Aquamex	55	490	502.6	+2.6	10	
K1	29	Aquamex	55	570	612.7	+7.5	plugged	
J1	31	Aquamex	55	650	708.4	+9.0	8	
I1	30	Aquamex	55	725	752.8	+3.8	>8	
H1	44	Aquamex	55	800	840.5	+5.1	17	
L2	34	Aquamex	55	875	925.8	+5.8	12	
G1	32	Aquamex	55	950	980.6	+3.2	10	Charge dislodged
K3	33	Aquamex	55	950	989.2	+4.1	>7	
L1	30	Aquamex	55	950	1000.5	+5.3	>1	Charge dislodged
F2	33	Aquamex	55	1025	1071.8	+4.6	10	
I3	32	Aquamex	55	1025	1078.1	+5.2	9	
F1	46	Aquamex	55	1125	1172.9	+4.3	13	
G3	32	Aquamex	55	1125	1184.9	+5.3	>7	
L3	37	Aquamex	55	1125			-3	misfire
F4	44	Aquamex	55	1225			11	misfire
E2A	34	Aquamex	55	1350	1337.3	-0.9	plugged	
H4	36	Aquamex	55	1350	1483.7	+9.9	9	
J4	37	Aquamex	55	1500	1536.9	+2.4	10	
G4	62	Aquamex	82.5	1675	1782.6	+6.3	20	
G4	52	Aquamex	82.5	1675	2311.1	+38.0		Very late detonation

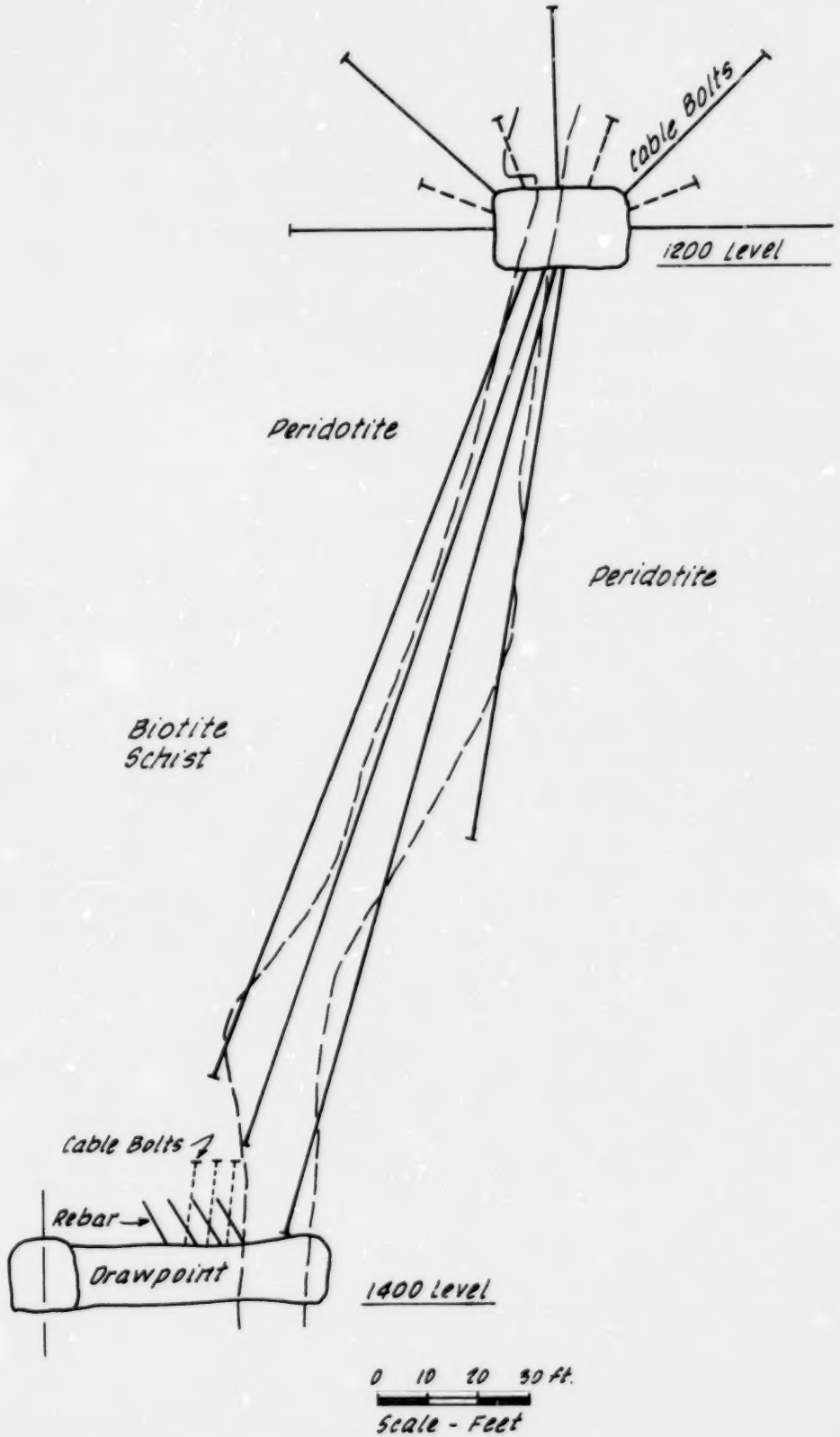
TABLE 4–12

BLAST DETAILS AND DIAGNOSIS, DECK BLAST

HOLE	CHARGE DEPTH (FT)	EXPLOSIVE TYPE	CHARGE WEIGHT (LB)	NOMINAL DELAY (MS)	ACTUAL DETONATION (MS)	DELAY ERROR (%)	BLAST INTERPRETATION COMMENTS
H2	13	Aquamex	82.5	230	250.4	+8.7	
	23	Aquamex	82.5	230	245.3	+10.6	Possible sympathetic
J3	11	Aquamex	82.5	280	288.6	+3.1	
	21	Aquamex	82.5	280	296.3	+5.8	
I1	19	Aquamex	82.5	340	347.4	+2.2	
I3	16	Aquamex	82.5	410	403.1	-1.7	
G3	18	Aquamex	55	490	505.6	+3.6	Low detonation efficiency
	28	Aquamex	55	490	511.4	+4.8	Low detonation efficiency
H4	28	Aquamex	82.5	570	614.0	+7.7	
L2	13	Aquamex	55	570	601.8	+5.6	Registered >1600 mm/s at B5 gauge
	23	Aquamex	55	570	604.8		Possible sympathetic detonation
H1	10	Aquamex	55	650	708.6		
	28	Aquamex	55	650	731.5		
J4	8	Aquamex	55	725	763.7		
	18	Aquamex	55	725	769.5		Dislodged charge
	28	Aquamex	55	725			Failed to detonate
I4	25	Aquamex	55	800	832		
	35	Aquamex	82.5	800	846		Allocation of vibration packets to precise
	45	Aquamex	82.5	800			decks not possible from here on
F2	4	Aquamex	55	800			Individual vibration packets identified
	14	Aquamex	55	800			3 X 800 ms decks misfired
	24	Aquamex	55	875	930		4 X 875 ms decks misfired
G4	18	Aquamex	82.5	875	944		
	28	Aquamex	82.5	875			Failed to detonate
	38	Aquamex	55	875			
F1	14	Aquamex	55	875			Charge malfunctions at 875 ms assigned
	24	Aquamex	55	875			to hole F1; caused by ground dislocation
	34	Aquamex	55	950	973		
L4	10	Aquamex	55	950	1025		
	20	Aquamex	55	950			Failed to detonate
	30	Aquamex	82.5	950			Failed to detonate
	40	Aquamex	82.5	950			Charge malfunctions in hole E4
E4	15	Aquamex	55	950			attributed to ground dislocation
	25	Aquamex	55	950			
	35	Aquamex	55	950			
	45	Aquamex	82.5	950			
K4	20	Aquamex	55	1025	1081		
	30	Aquamex	55	1025	1090		
	40	Aquamex	82.5	1025	1105		
	50	Aquamex	82.5	1025			Failed to detonate
L1	14	Aquamex	55	1025			Failed to detonate
	24	Aquamex	55	1025			Failed to detonate
D2	30	Aquamex	55	1025			Charge malfunctions in hole D2
	40	Aquamex	55	1025			attributed to ground dislocation
	50	Aquamex	55	1025			
	60	Aquamex	55	1025			
	70	Aquamex	55	1025			
	80	Aquamex	82.5	1025			
H5	12	Aquamex	55	1125	1159		
	22	Aquamex	55	1125	1171		
	32	Aquamex	82.5	1125	1189		
	42	Aquamex	82.5	1125			Failed to detonate
	52	Aquamex	55	1125			Failed to detonate
D1	10	Aquamex	55	1125			Charge malfunctions in hole D1
	20	Aquamex	55	1125			attributed to ground dislocation
	30	Aquamex	55	1125			
	40	Aquamex	55	1125			
	50	Aquamex	55	1125			
	60	Aquamex	82.5	1125			
D3	15	Aquamex	55	1225	1287		Low amplitude, 20 mm/s
	25	Aquamex	55	1225			
	35	Aquamex	55	1225			
	45	Aquamex	55	1225			
	55	Aquamex	55	1225			
	65	Aquamex	82.5	1225			
D4	15	Aquamex	55	1400	1460		15mm/s
	25	Aquamex	55	1400	1507		7mm/s
	26	Aquamex	55	1400			
	37	Aquamex	82.5	1675	1794		7mm/s
C2	15	Aquamex	55	1675			
	24	Aquamex	55	1950	2166		8mm/s
	35	Aquamex	55	1950	2260		22mm/s
	45	Aquamex	82.5	1950	2319		24mm/s
C1	7	Aquamex	55	1950			
	17	Aquamex	55	1950			
	27	Aquamex	82.5	1950			
C3	11	Aquamex	55	2275	2496		4mm/s
	21	Aquamex	55	2275	2546		10mm/s
	31	Aquamex	55	2275			
	41	Aquamex	82.5	2275			

THOMPSON MINE 3 SHEAR TRANSVERSE SECTION

Figure **4–1**



PROJECT NO. 852-1605 DRAWN [illegible] REVIEWED ..... DATE Sept. 89

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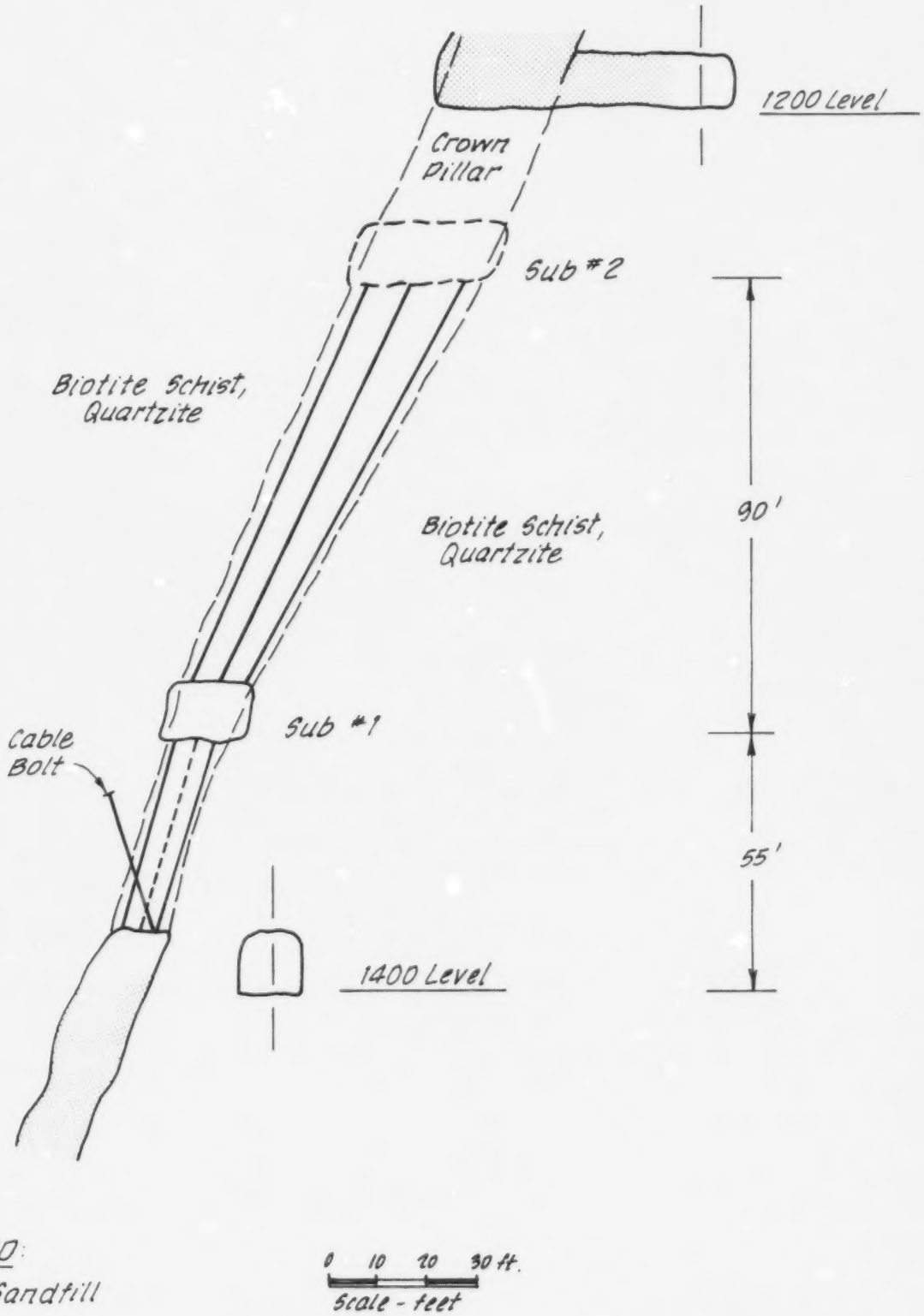
THOMPSON MINE 1400 -330 STOPE
TRANSVERSE SECTION

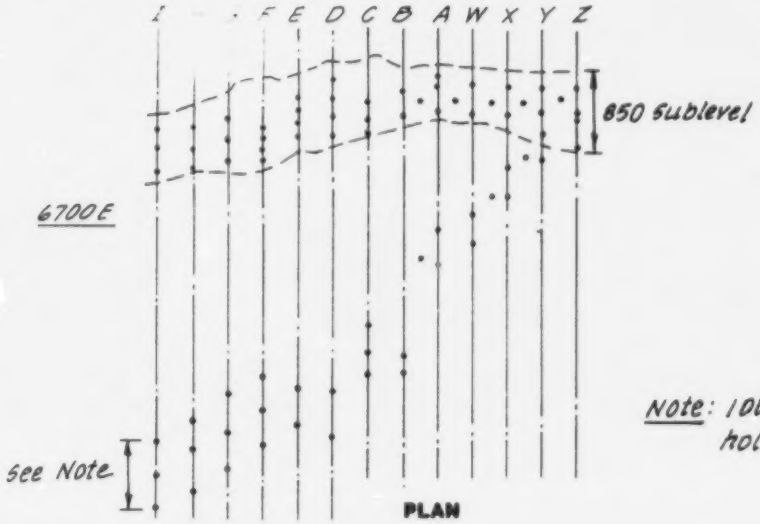
Figure **4-2**



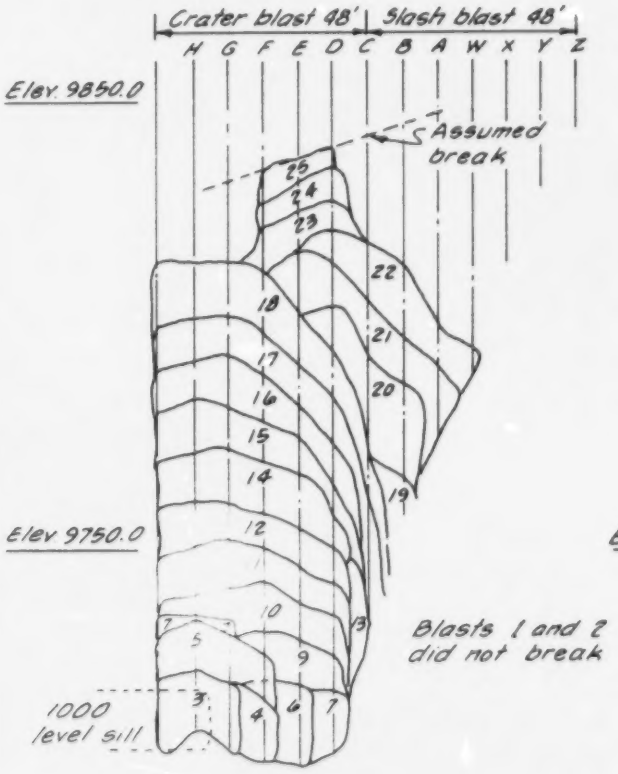
PROJECT NO 832-1605 DRAWN [illegible] REVIEWED DATE Sept 89

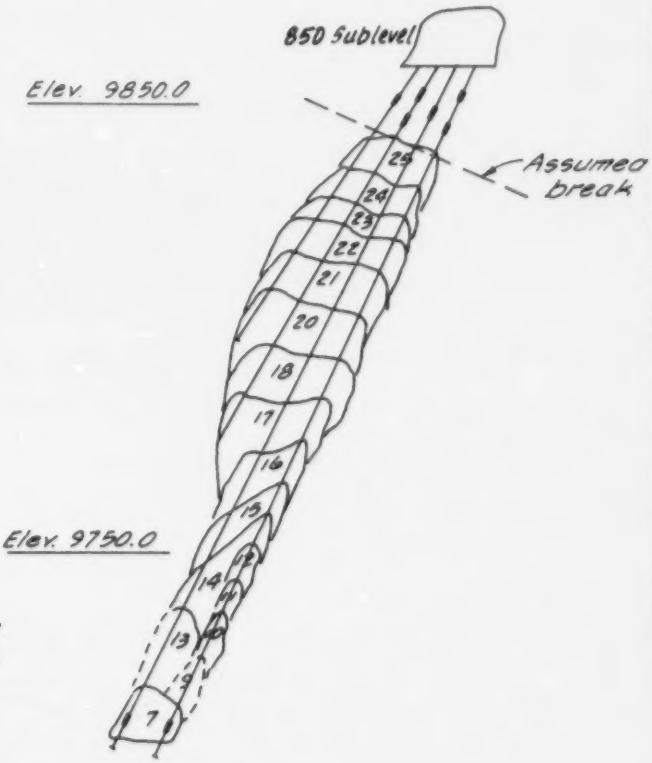
3 SHEAR VBM BLOCK 1
850-1000 LEVEL PLAN AND SECTION
VIEWS SHOWING EXTRACTION SEQUENCE

Figure **4-3**



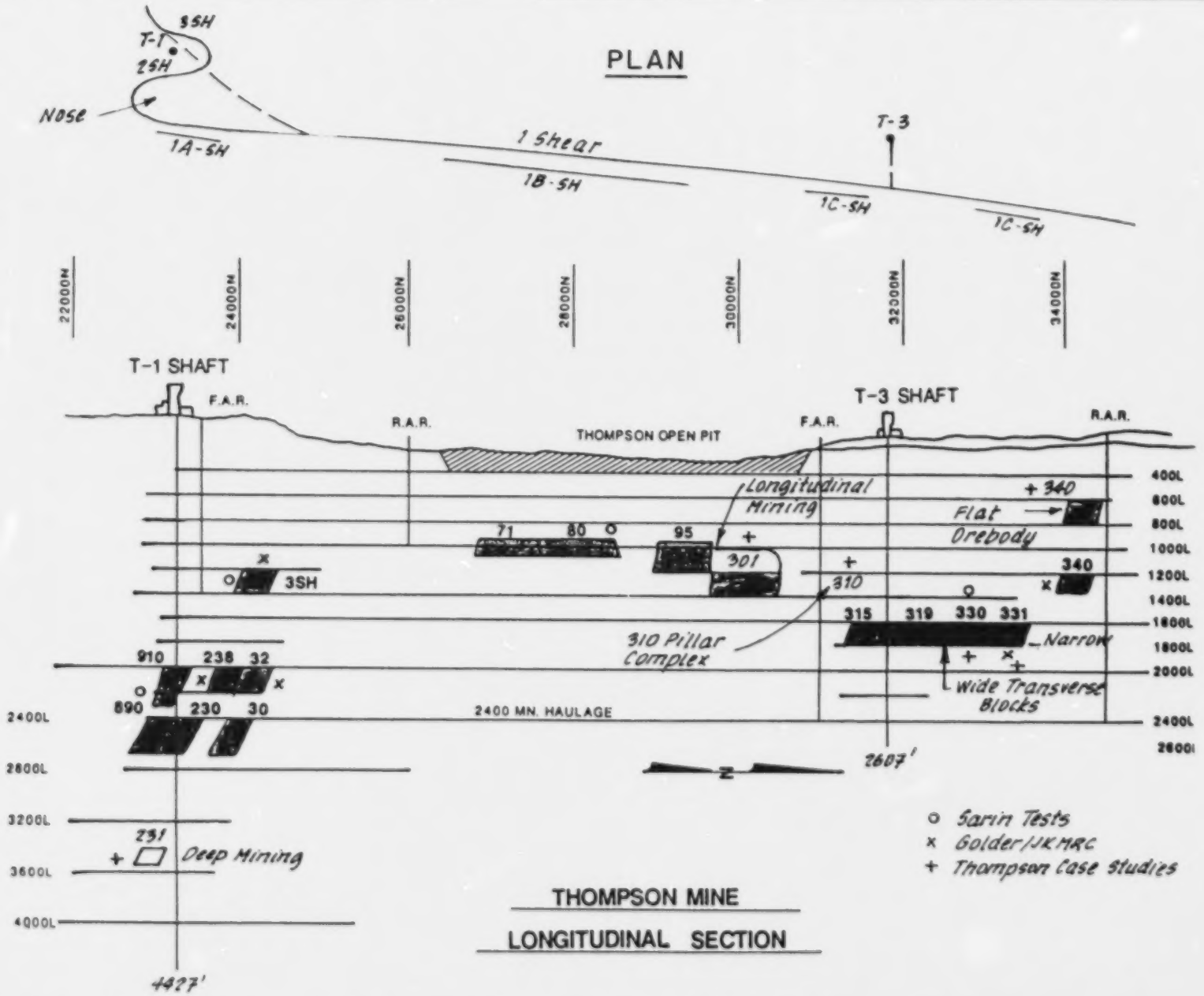
Note: 1000 Level break through holes (rows I and H).


Crater blast 48'
Slash blast 48'
H G F E D C B A W X Y Z
Elev. 9850.0
Assumed break
Elev. 9750.0
1000 level sill
Blasts 1 and 2 did not break
LONGITUDINAL VIEW



PROJECT NO 852-1605 DRAWN REVIEWED DATE 1 of 1 85

PROJECT NO 852-1605 DRAWN REVIEWED DATE Sept. '89



THOMPSON MINE

LONGITUDINAL SECTION

Figure 4-4

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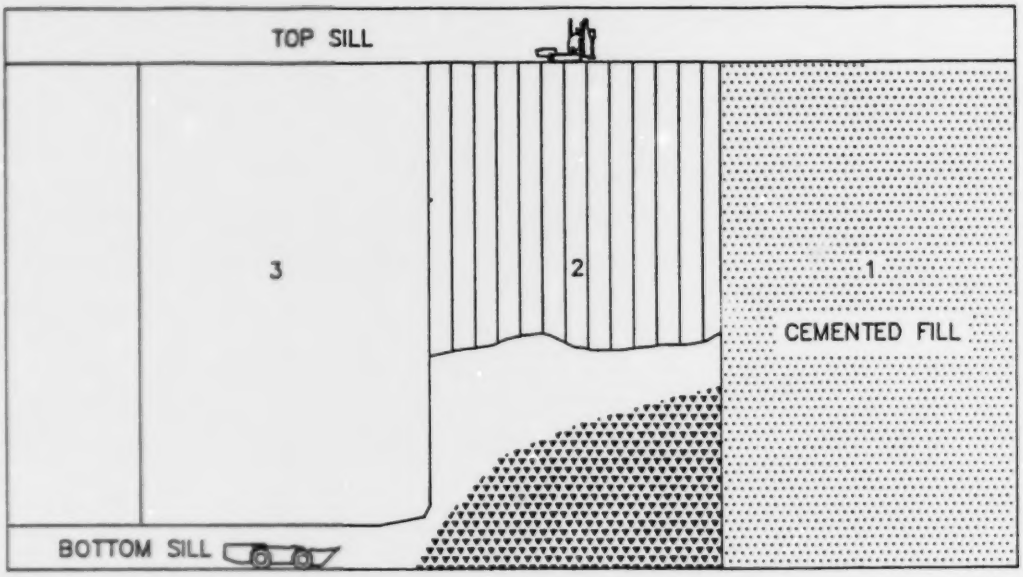
VBM - LONGITUDINAL MUCKING

Figure 4-5

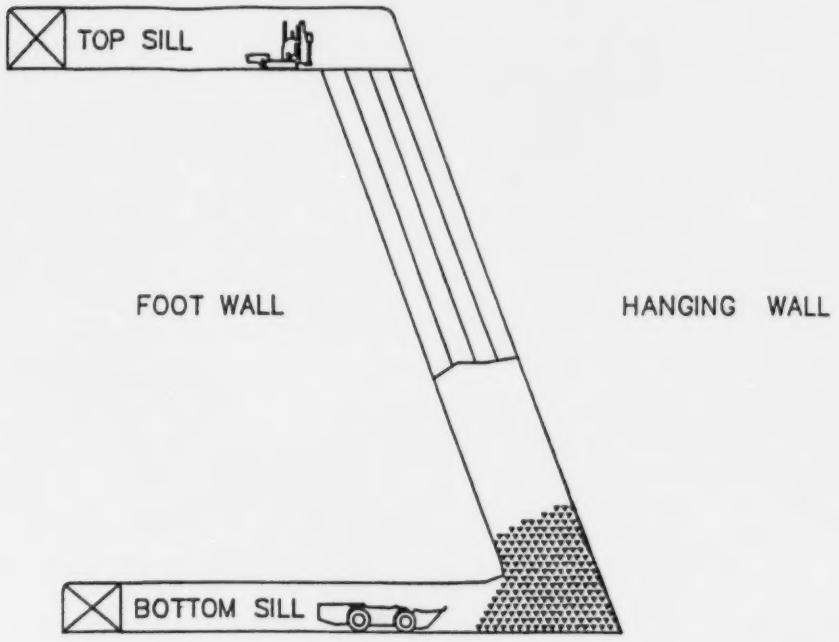


Not to scale.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

VBM - TRANSVERSE MUCKING

Figure **4-6**

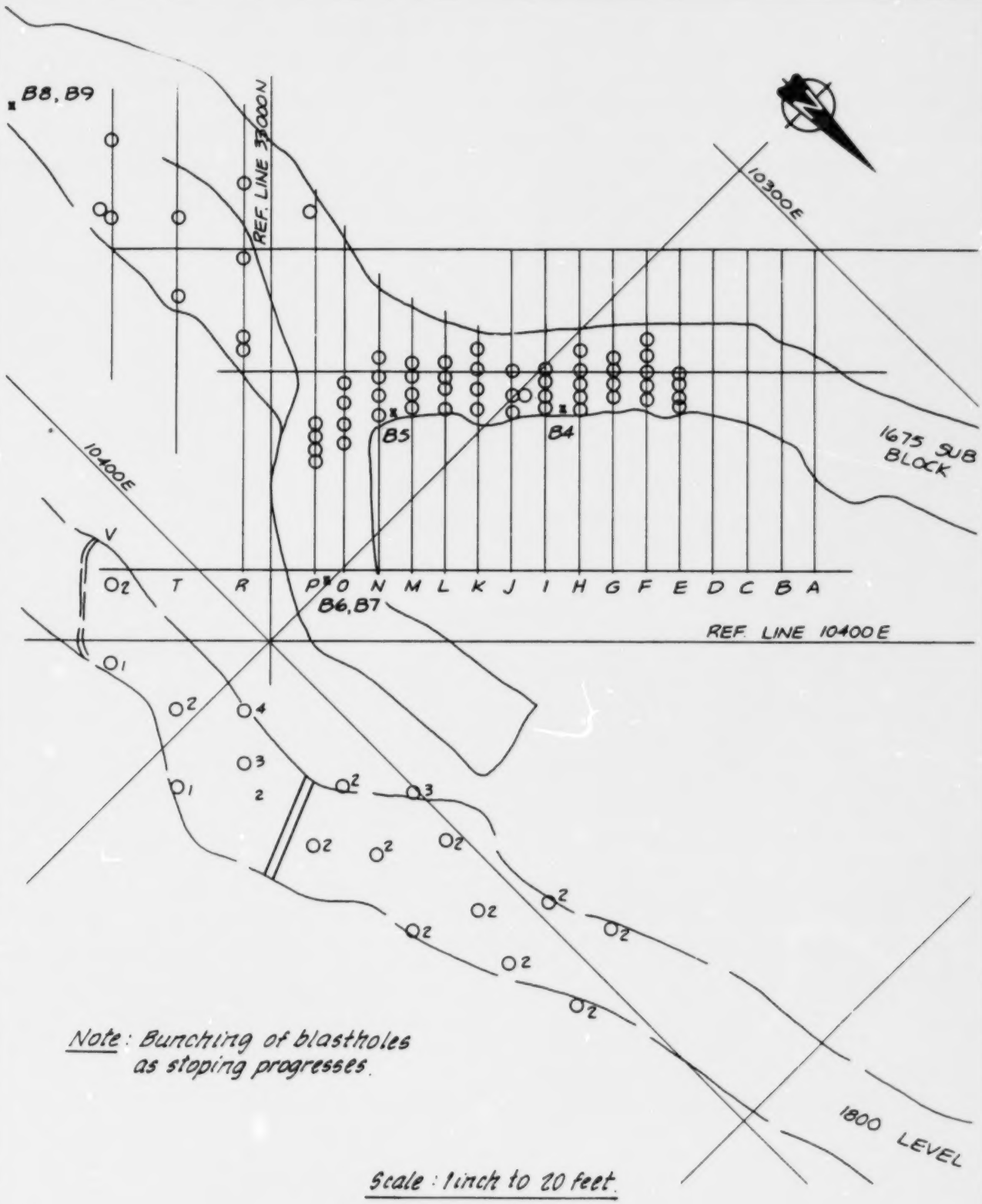


Not to Scale.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

1803 - VBM STOPE #4 BLOCK
PLAN VIEW 1675 DRILLING LEVEL

Figure **4-7a**



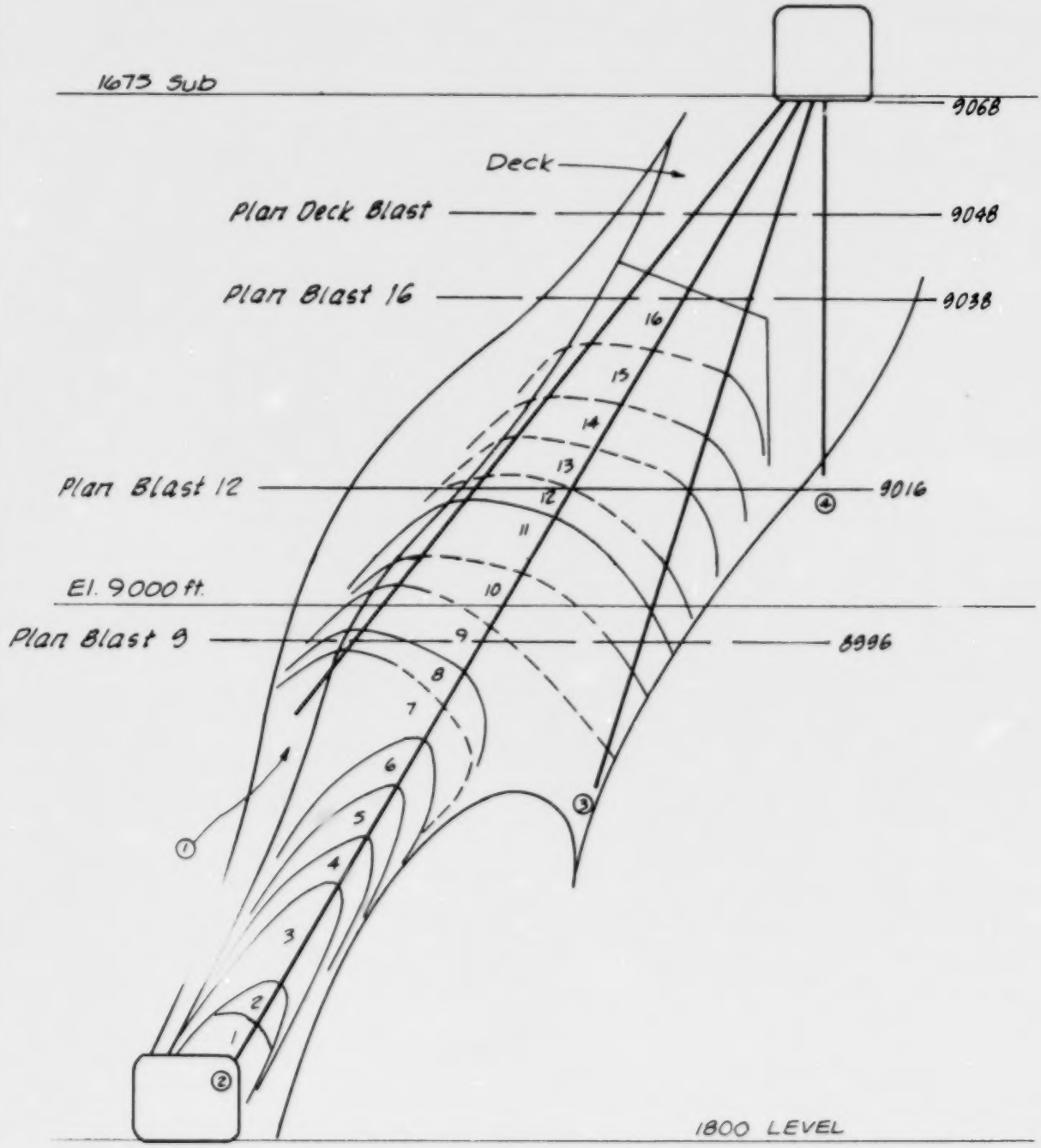
Note: Bunching of blastholes as stoping progresses.

Scale: 1 inch to 20 feet.

PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

1803 - 331 VBM STOPE #4 BLOCK BREAKAGE PROFILE, TRANSVERSE SECTION THROUGH ROW I (LOOKING SOUTH)

Figure **4–7b**



PROJECT NO DRAWN REVIEWED DATE

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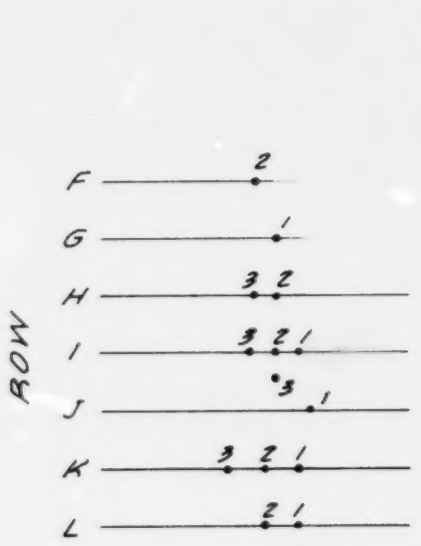
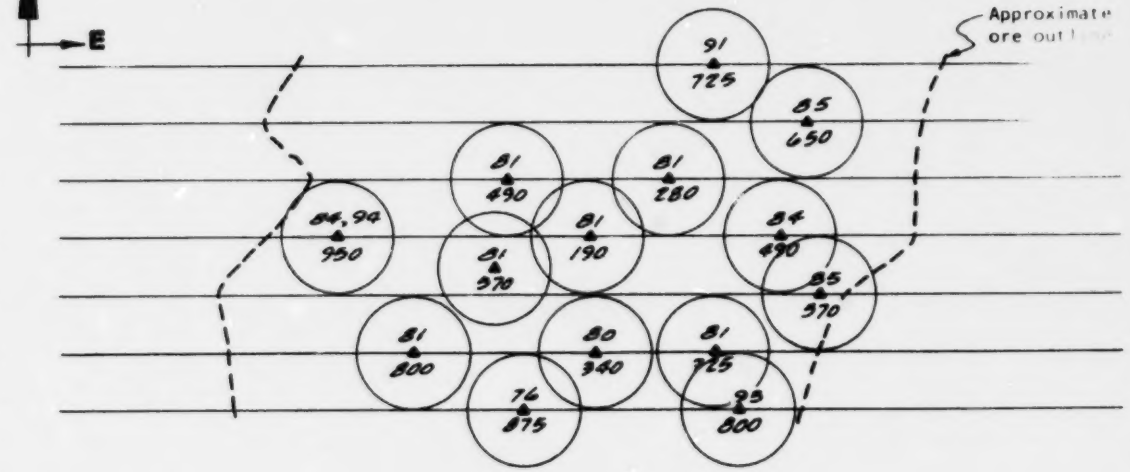
PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....



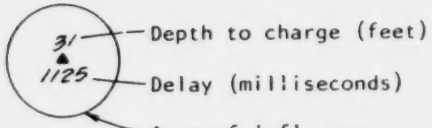
PLAN OF HOLE COLLAR
ELEVATION 9068 ft.

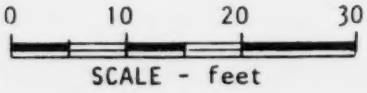


PLAN AT ELEVATION 8996 ft.

HOLE NUMBERS	HOLE TO HOLE DISTANCES (feet)	CHARGE TO CHARGE DISTANCES (feet)
I1 to J1	5.9	5.1
I1 to H2	11.1	6.8
I1 to K1	11.7	11.3
H3 to I2	8.6	5.7
H3 to J3	8.1	9.3

LEGEND





EXPLOSIVE DISTRIBUTION
1803-331 VBM BLOCK #4 BLAST #9

Figure 4-7C

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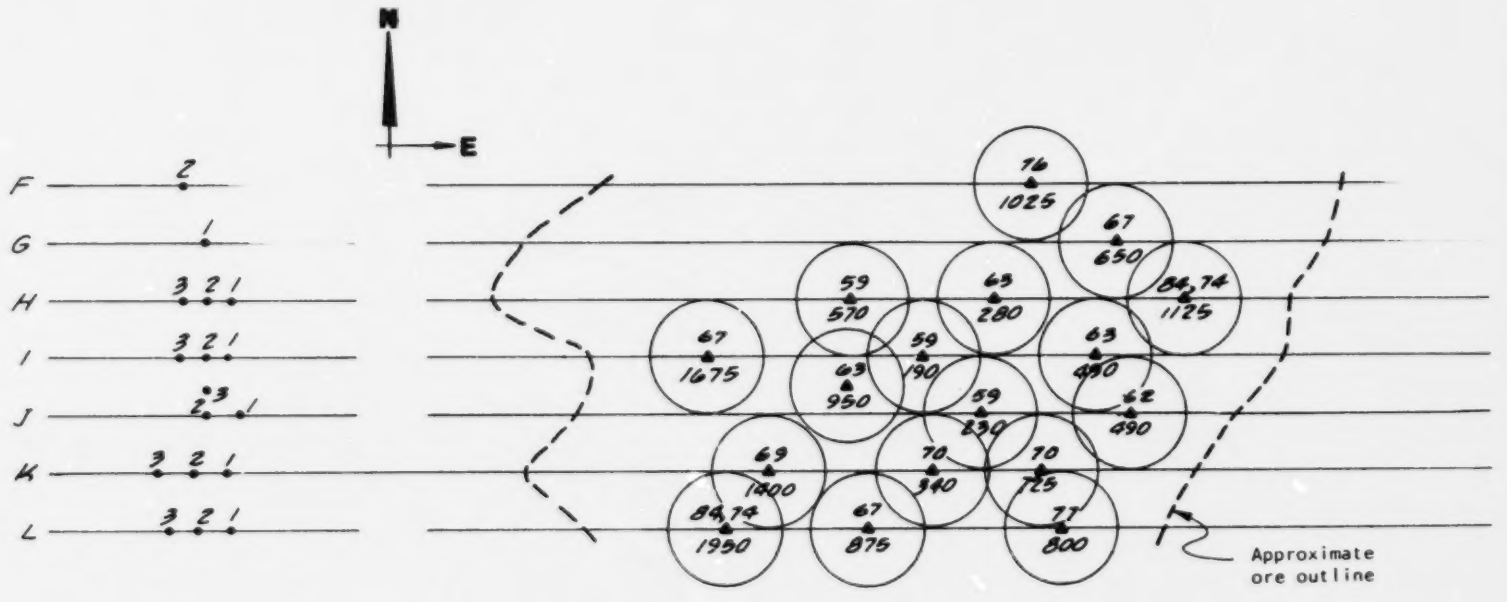
PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

EXPLOSIVE DISTRIBUTION
1803-331 VBM BLOCK #4 BLAST #12

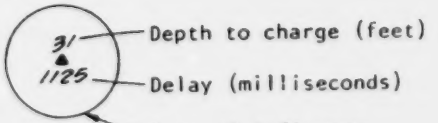


PLAN OF HOLE COLLAR ELEVATION 9068 ft.

PLAN AT ELEVATION 9016 ft.

HOLE NUMBERS	HOLE TO HOLE DISTANCES (feet)	CHARGE TO CHARGE DISTANCES (feet)
12 to J2	6.8	5.9
12 to H2	7.9	7.7
12 to J3	7.6	7.3
12 to H3	8.8	5.3
13 to J3	12.5	9.1

LEGEND



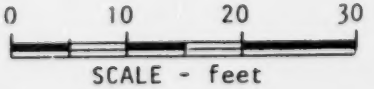


Figure 4-7d

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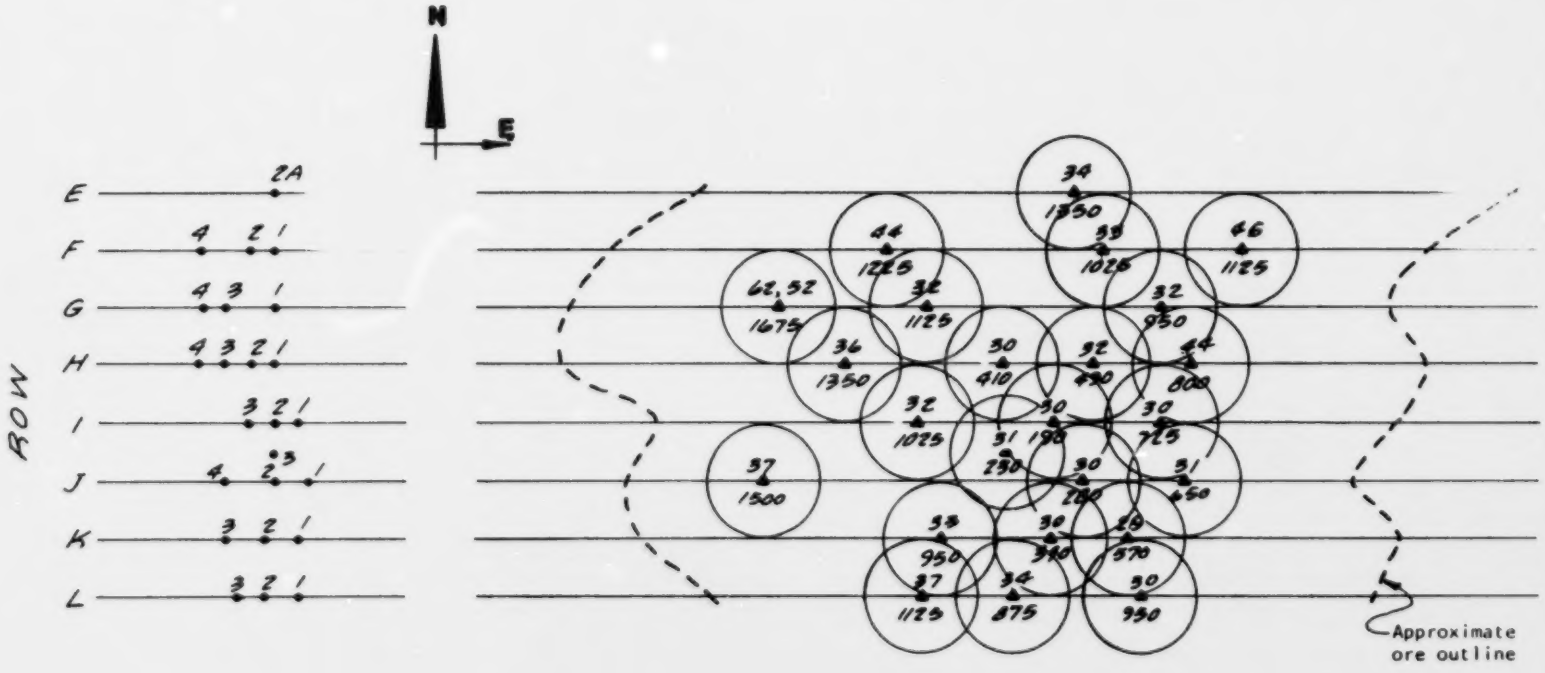
PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

EXPLOSIVE DISTRIBUTION
1803-331 VBM BLOCK #4 BLAST #16

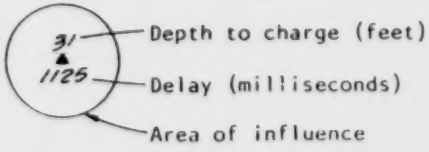


PLAN OF HOLE COLLAR ELEVATION 9068 ft.

PLAN AT ELEVATION 9038 ft.

HOLE NUMBERS	HOLE TO HOLE DISTANCES (feet)	CHARGE TO CHARGE DISTANCES (feet)
I2 to J3	5.0	4.1
I2 to H2	6.1	5.8
G4 to J4	15.0	22.3
F4 to G3	6.1	13.7
E2A to F2	5.6	6.8

LEGEND



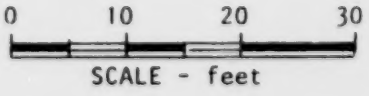


Figure 4-7e

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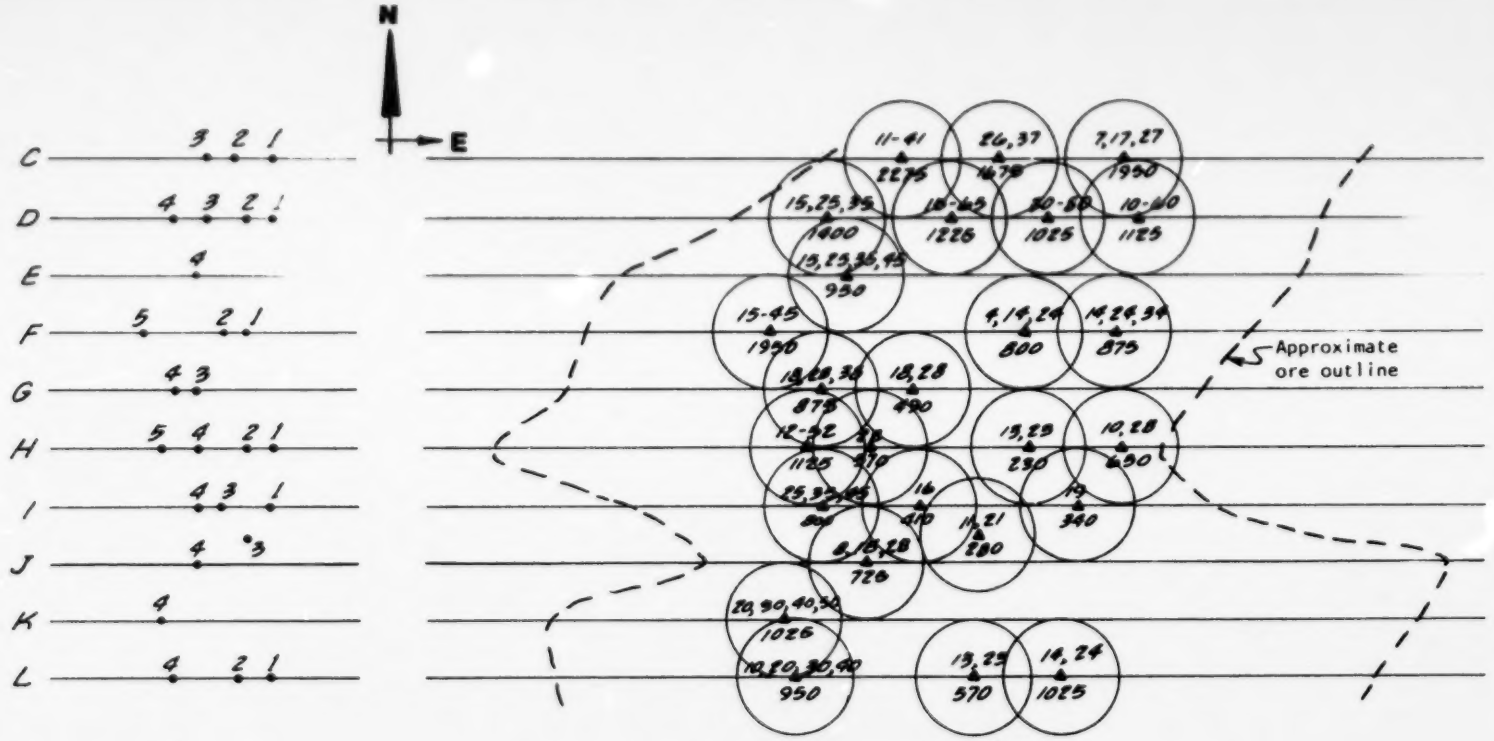
PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....

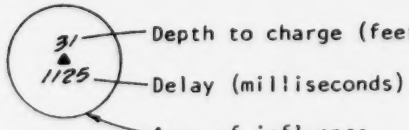


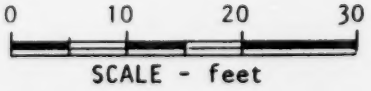
PLAN OF HOLE COLLAR ELEVATION 9068 ft.

PLAN AT ELEVATION 9048 ft.

HOLE NUMBERS	HOLE TO HOLE DISTANCES (feet)	CHARGE TO CHARGE DISTANCES (feet)
G3 to H4	6.3	7.0
G3 to E4	11.7	12.8
G3 to F2	10.8	10.7
D2 to D3	8.4	10.2
D2 to F2	10.2	12.4

LEGEND





EXPLOSIVE DISTRIBUTION
1803-331 VBM BLOCK #4 DECK BLAST

Figure 4-7f

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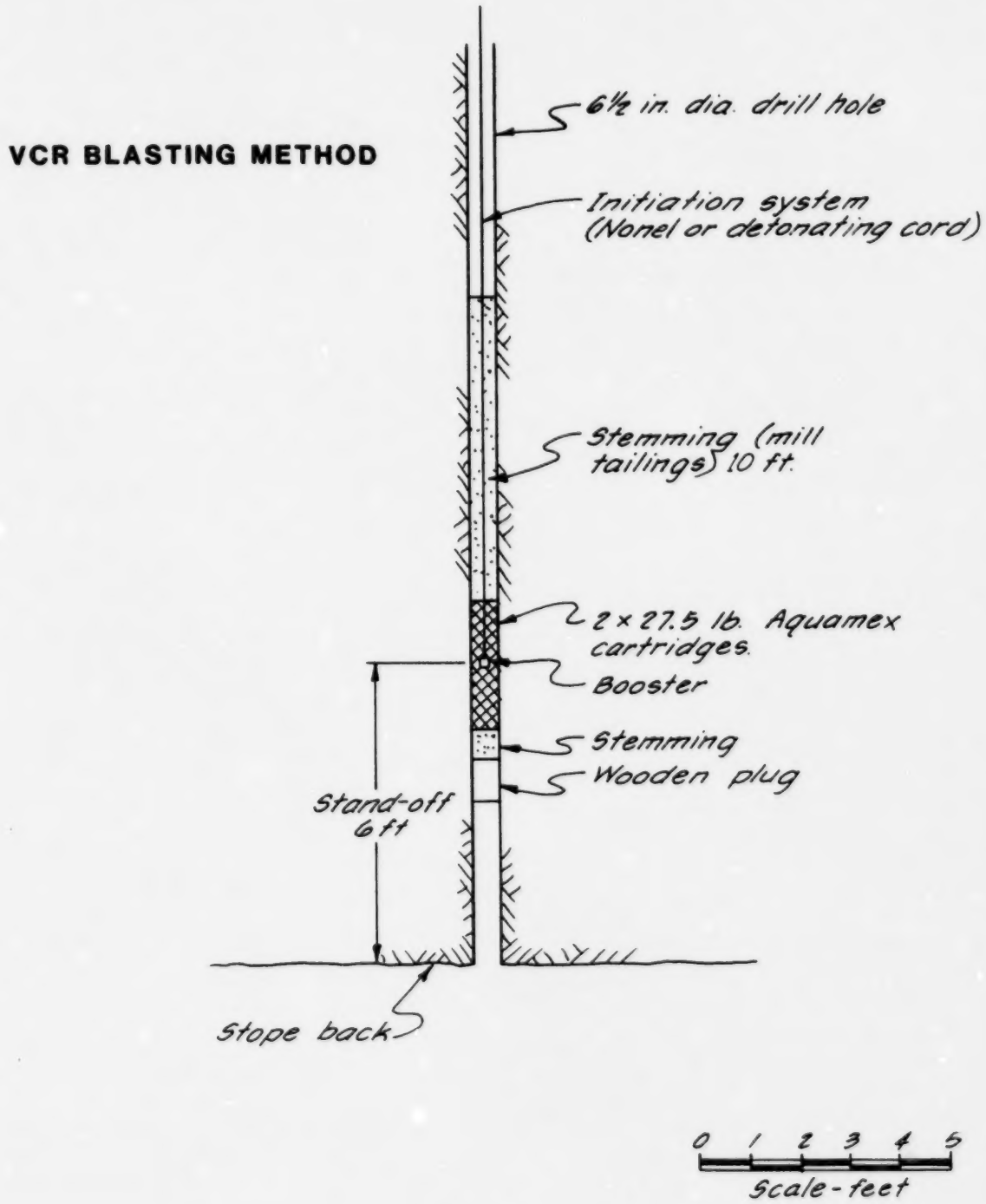
CHARGE CONFIGURATION AND GEOMETRY FOR 6.5 inch BLASTHOLE

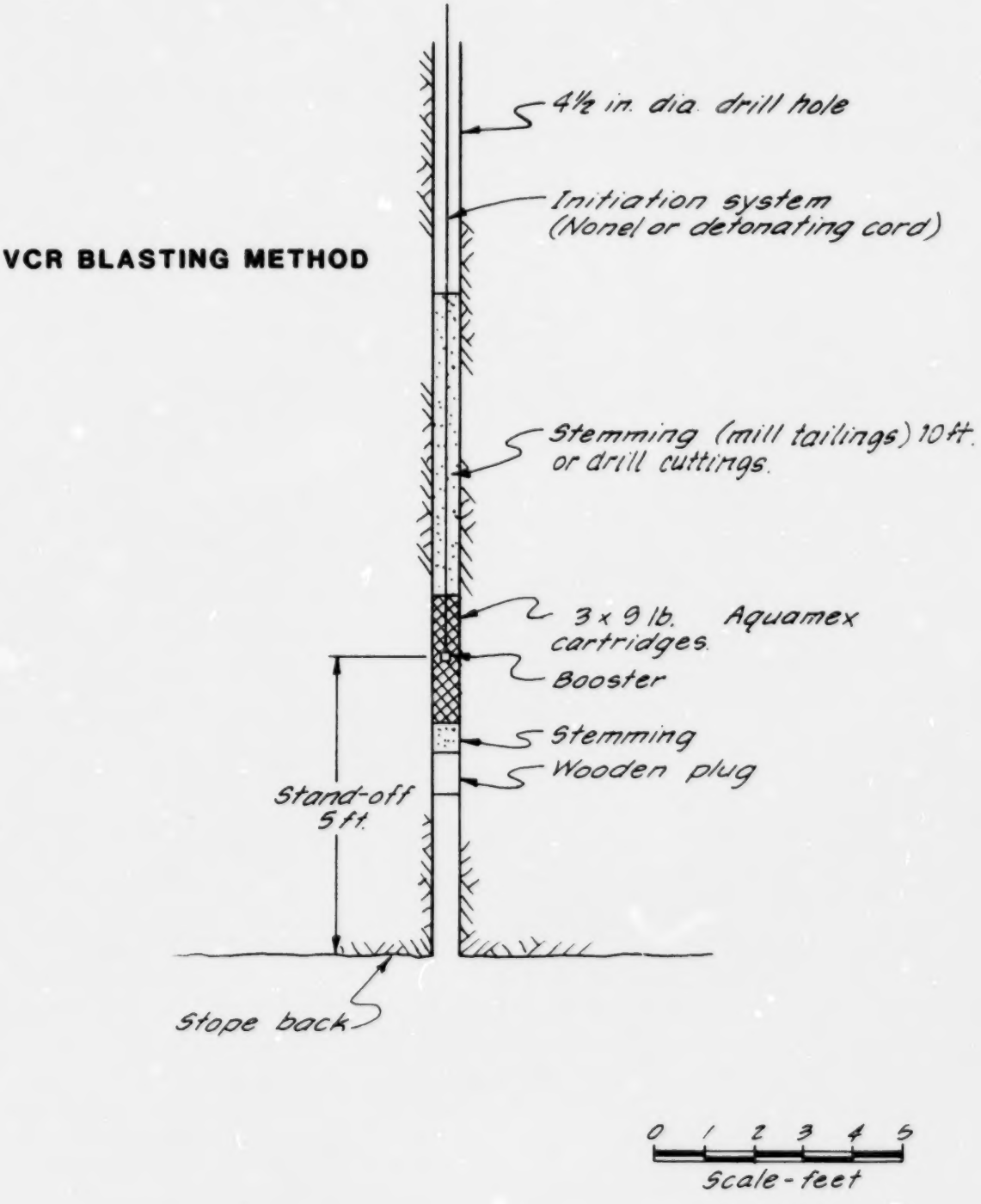
Figure **4-8**



PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

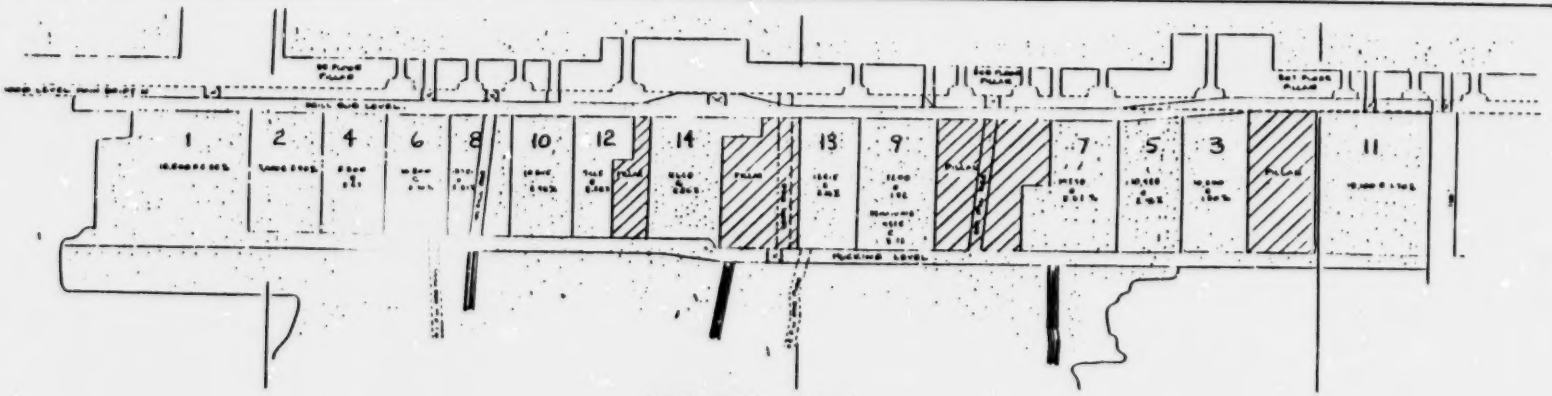
CHARGE CONFIGURATION AND GEOMETRY FOR 4.5 Inch BLASTHOLE

Figure **4-9**

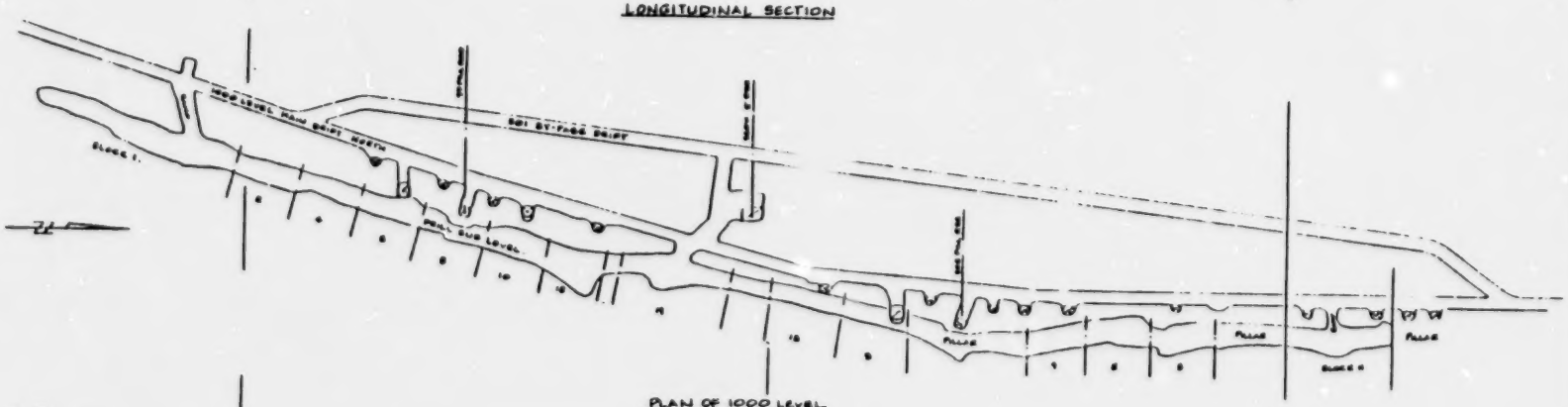


PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

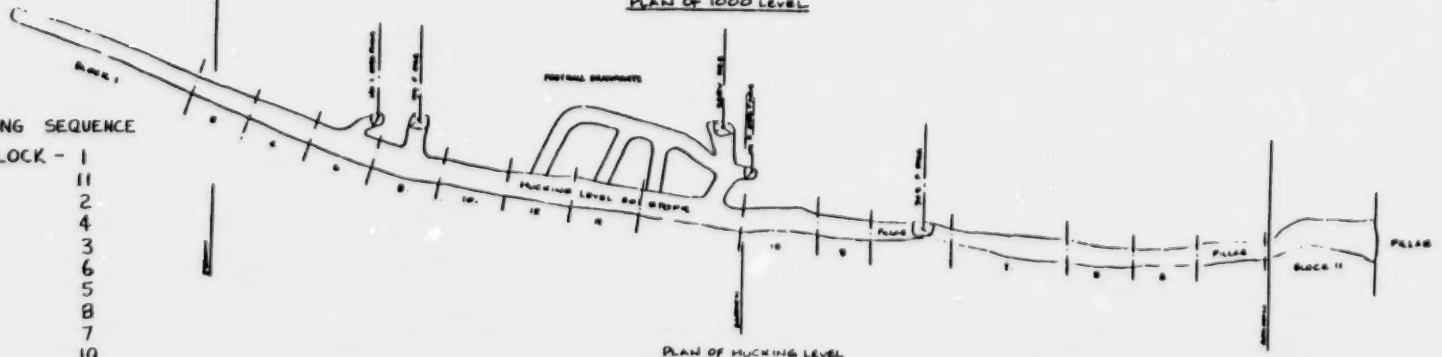
PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....



LONGITUDINAL SECTION



PLAN OF 1000 LEVEL



PLAN OF MUCKING LEVEL
801 STOPE

MINING SEQUENCE

BLOCK - 1
11
2
4
3
6
5
8
7
10
9
13
12
14

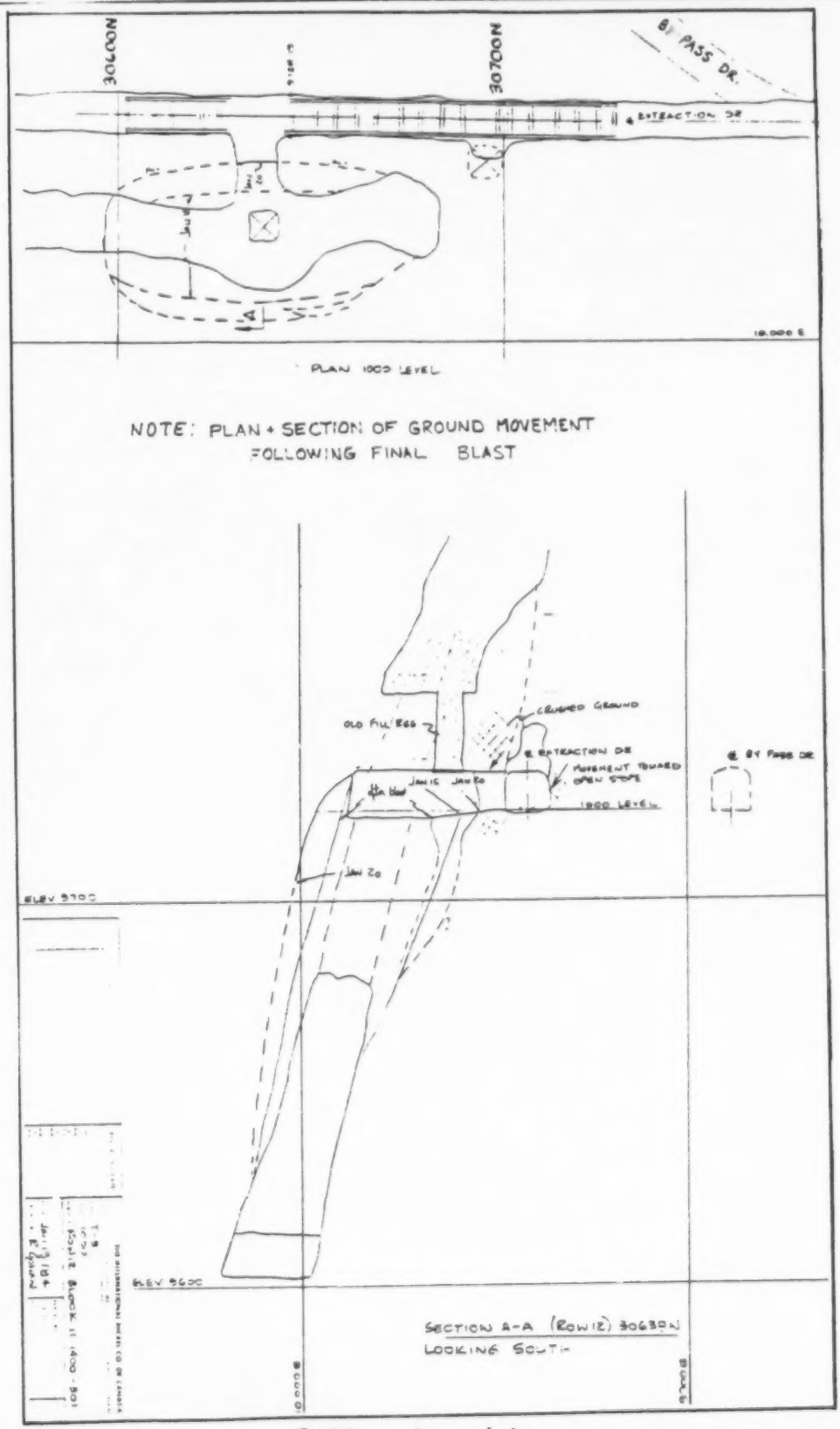
1403 LEVEL 301 VBM STOPE
LONGITUDINAL SECTION & COMPOSITE PLAN

Figure 4-10

Golder Associates

1403-301 VBM STOPE BLOCK II

Figure **4-11a**



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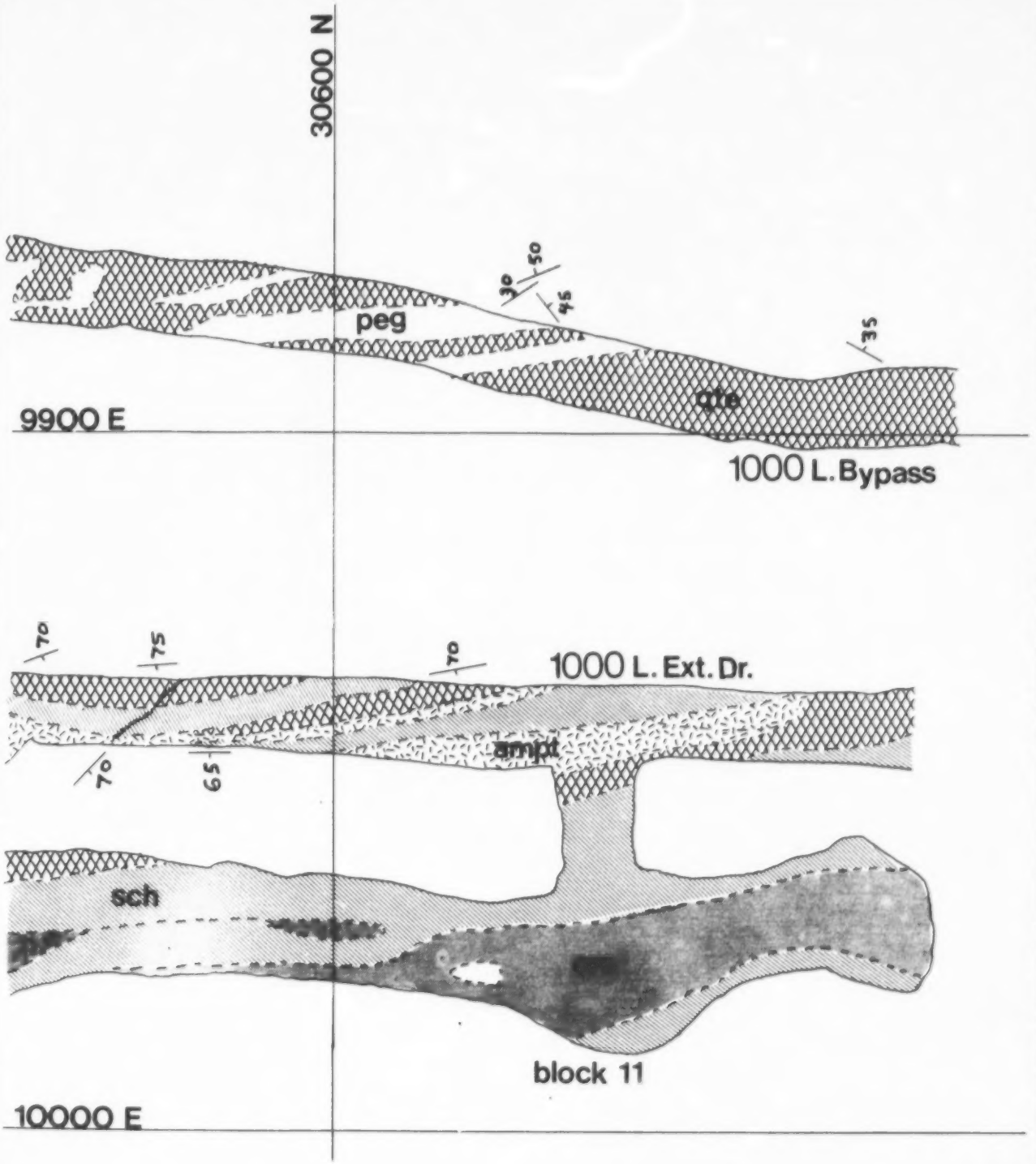
GEOLOGY 1403-301 VBM STOPE BLOCK 11

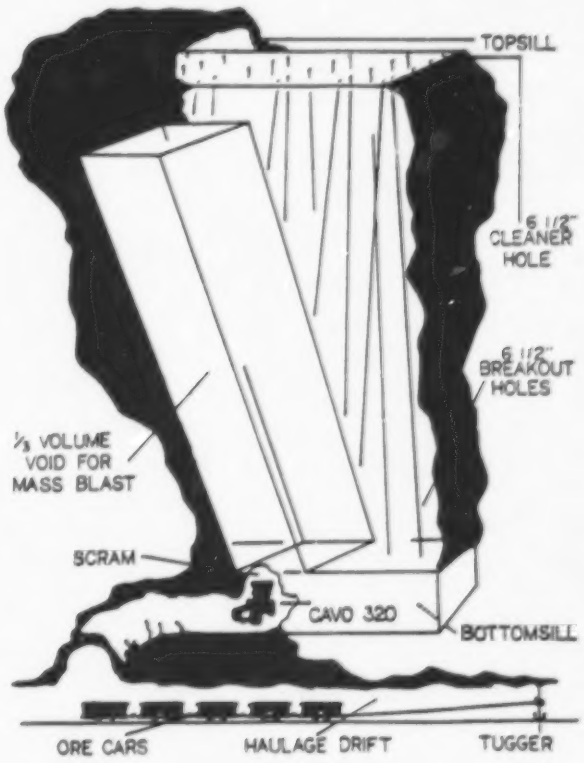
Figure 4-11b



PROJECT NO DRAWN REVIEWED DATE

DEPICTION OF SMALL BLOCK VCR STOPING AT HOMESTAKE

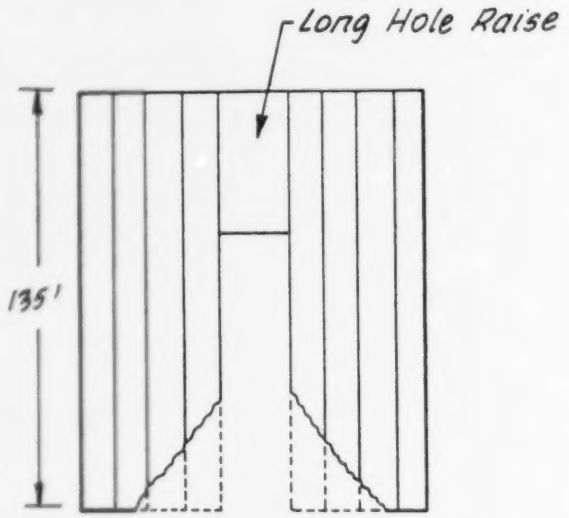
Figure **4-12**



PROJECT NO DRAWN REVIEWED DATE

LONG SECTION SHOWING SLOUGHING OF THE SLOT AT COBAR

Figure **4–13**



Note: Possible failure of ground due to raise and slot development at stope centre.

Not to Scale.

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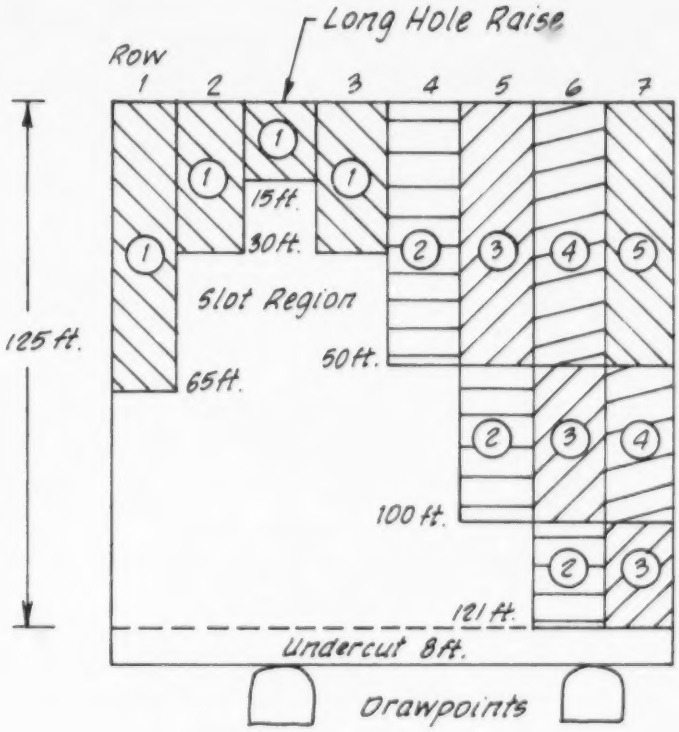
BLASTING SEQUENCE FOR UNDERHAND STOPING AT COBAR

Figure **4-14**

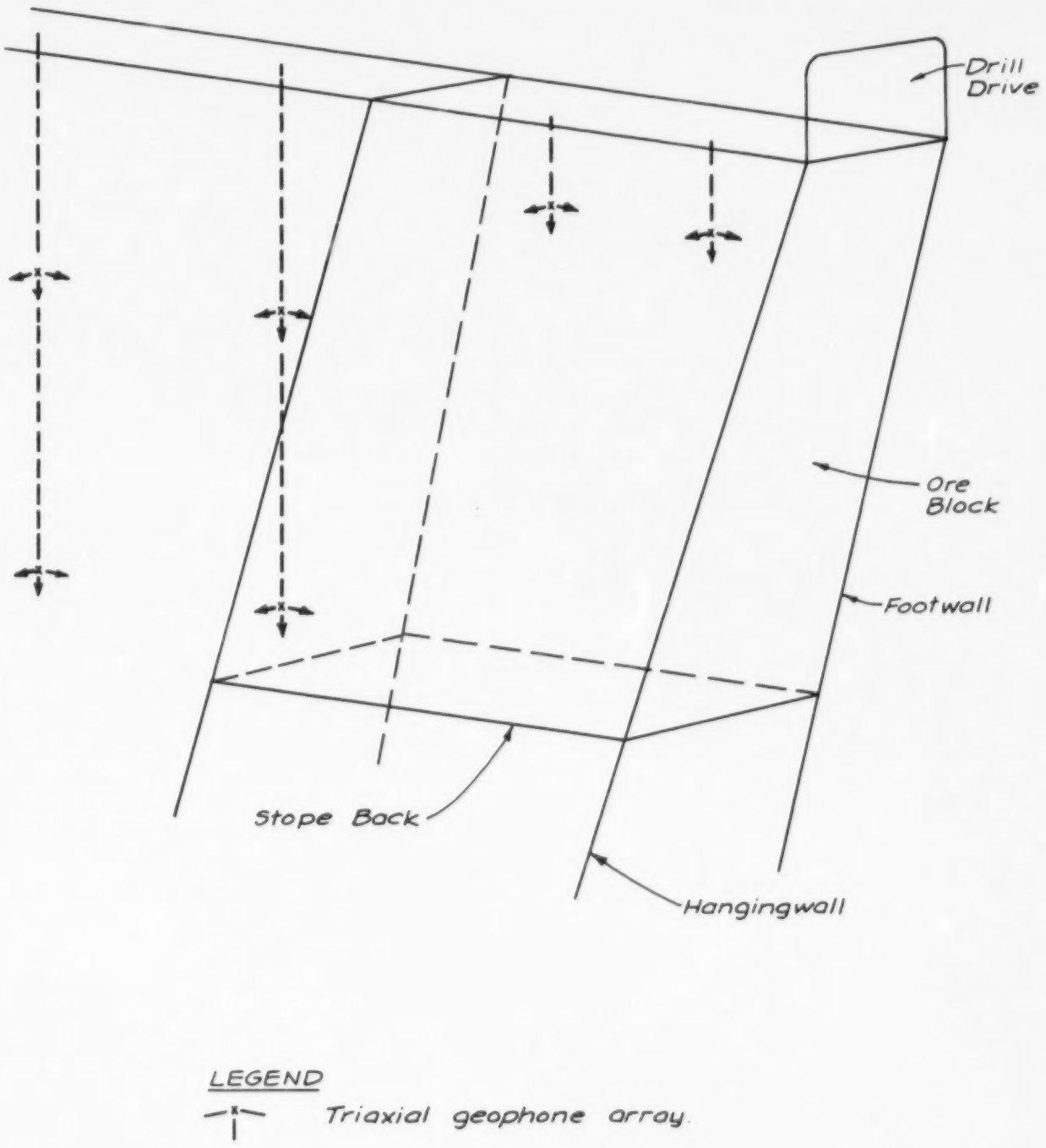


Long Section
Not to Scale.

PROJECT NO. ..... DRAWN ..... REVIEWED ..... DATE .....

VBM STOPE MONITORING ARRANGEMENT

Figure **4–15**



PROJECT NO ..... DRAWN ..... REVIEWED ..... DATE .....

APPENDIX 4-A

VERTICAL CRATER
RETREAT BLASTING

VERTICAL CRATER RETREAT BLASTING

by

G.G. PAINE[1], G. HARRIES[2] AND T.N. HAGAN[3]

(Extract from Australian Mineral Foundation and Blasting Workshop, Chapter 3, 1979).

Section	Topic
3.1	INTRODUCTION
3.2	CRATER THEORY
3.3	EXPLOSIVES FOR VCR
3.4	CRATER TESTING
3.5	CHARGING TECHNIQUES FOR PRODUCTION BLASTHOLES
3.6	PRACTICAL CONSIDERATIONS
3.7	BLAST RECORDS
3.8	REFERENCES

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1. G.G. Paine, Senior Explosives Field Engineer, Nobel Group, ICI Australia Operations Pty Limited, Brisbane.

2. G. Harries, Senior Blasting Physicist, Nobel Group, ICI Australia Operations Pty Limited, Melbourne.

3. T.N. Hagan, Senor Mining Engineer (Applied Blasting Research), Nobel Group, ICI Australia Operations Pty Limited, Melbourne.

VERTICAL CRATER RETREAT BLASTING

by

G.G. PAINE[1], G. HARRIES[2] and T.N. HAGAN[3]

3.1 Introduction

The introduction of large (i.e. 102-200 mm (4 in.-8 in.)) diameter blastholes to underground mining operations has allowed the development and use of the Vertical Crater Retreat (VCR) mining method. VCR is more suited to:

1. narrow, near vertical orebodies;
2. large orebodies with weak hangingwalls and/or footwalls;
3. pillar recovery, especially in the situations in 1 and 2 above;
4. the mining of shallow orebodies from the surface;
5. raising; and
6. slot development.

Large diameter blastholes which are vertical or near vertical are drilled either from a drive (in the case of downhole fans) or from a topsill (in the case of parallel blastholes). Some of these blastholes must break through into the undercut.

Charges with a length:diameter ratio <6, these being equivalent to and termed spherical charges, are positioned at the calculated optimum distance from the back of the undercut.

A sequence of blastholes in a blast break and displace a horizontal slice of rock which falls downwards into the previously mined area. By repeating this charging and firing procedure, mining advances upwards in successive horizontal slices.

When applied to primary stopes and pillar recovery, the VCR method:

1. eliminates raise boring and slot development;
2. generally gives improved fragmentation;
3. permits shrinkage draw of the broken rock.

3.2 Crater Theory

When a spherical or short cylindrical charge of explosive is buried at less than the critical depth (d_c) below the surface and detonated, a crater if formed. A typical experimental curve showing the effect of depth of burial of charge on crater volume is shown in Figure A.1. The depth of burial which gives the largest crater is known as the optimum depth (d_o).

At the critical depth of burial no crater is formed, but cracks are just visible around the collar of the blasthole. As the stemming is usually found in place, the explosion gases have not escaped to the surface. The depth d_c, therefore, represents the distance to which explosive shock induced cracks can propagate. As the depth of burial is reduced, ground movement can take place. This enables the gas to enter the cracks and heave the rock to form narrow craters. Further reductions in the depth of burial give larger, wider craters, the gas being able to penetrate into and extend more cracks.

When the burden is equal to the optimum burden, the longest cracks can just reach the surface. This gives the largest (i.e. optimum) crater. Harries (1977) has shown that the optimum crater and the fragmentation of the rock can be described by assuming that 7 cracks just reach the periphery of the optimum crater. The length (L) and the number of cracks (N) which have at least this length are given by:

$$N = N_0 \cdot \frac{b}{L} \cdot e^{-\alpha \frac{L}{b}} \qquad \ldots 3.1$$

where N_0 is the number of cracks at the periphery of the blasthole;

b is the radius of the blasthole, and

α is the strain absorption coefficient.

When the crack lengths given by the above distribution are placed uniformly around a charge, many of these cracks reach to a plane situated at about the optimum depth of burial from the charge (Harries, 1973). The maximum length to which the gas can extend the cracks is given by the slant height of the optimum crater. This is usually about twice the critical depth (Harries, 1977). The optimum crater gives a very good estimation of the maximum amount of work that can be done by a particular explosive in a particular rock (Livingston, 1956).

Decreasing the depth of burial further gives a smaller crater, since the gas can now escape at the free surface without doing any useful work. This creates flyrock and airblast. The optimum (d_o) gives the burden at which the explosive is most effective.

A reasonable description of cratering can be given by assuming that the charge is spherical and that the rock is strained equally in all directions. In practice, however, short cylindrical charges are used. The best description is given by incorporating the end effects from short cylindrical charges.

The theory shows that a true spherical charge in hard rock (basalt) will give a crater which subtends an angle of 110 degrees. Spherical charges are infrequently used in mining but have been used in the Plowshare

Nuclear Engineering programme. The large charges used in this programme give optimum craters which subtend an angle of 105 degrees (Nordyke, 1961, 1964).

Cylindrical charges can be described by splitting them into a series of spherical charges. The radius of each sphere is equal to 1.225 times the radius of the cylinder. This gives a series of spheres which just touch and have the same length as the cylinder. As the number of spheres increases and the cylindrical charge becomes longer, the radial strain increases and approaches a value equal to twice that of a single sphere. The radial strain from a sphere is about half that from the side of an infinitely long cylindrical charge (Harries, 1973). The ends of the cylindrical charge can not be approximated by hemispheres having a radius equal to 1.225 times the radius of the cylinder.

As the charge geometry is changed from a sphere to a squat cylinder, the strain in the plane parallel to the free face increases. This has the effect of broadening the crater. As most cratering tests are squat cylindrical charges, the craters should be broader than the craters from spherical charges. When the length:diameter ratio is 6, the calculated crater angle is 135 degrees. The average angle found in practice is 138 degrees (Harries, 1973). The radius of a true spherical charge with a length:diameter ratio of 6 is 2.08 times the radius of the cylinder. For a true spherical charge, therefore, the strain normal to the free face will be higher than that from the hemispherical end of the cylinder. The optimum depth of burial for a cylindrical charge will be less, but the crater will be wider than that created by the equivalent spherical charge. Detailed calculations for a range of cylindrical charges of different lengths show that spherical charges are the most effective. Short cylindrical charges with length:diameter ratios of >7 form good craters. Further increases in length show little increase in cratering ability, as more of the strain in directed parallel rather than normal to the free surface.

When using VCR changes as proposed by Lang (1978), arrays of charges are used. The optimum depth of burial can be calculated when the rock and explosive properties are known; for a particular explosive/rock combination, it can also be found by experiment. The spacing of the charges is controlled by the fragmentation required and by the necessity to maintain a straight back. To ensure these requirements are met, a detailed examination of crack propagation in three dimensions is necessary.

3.3 Explosives for VCR

For all but the weakest rock formations, there is a need to maximize the effective explosives energy generated per unit length of charge. This requirement is generally met by using an explosive which exhibits:

1. high velocity of detonation;
2. high density; and
3. the ability to fill the entire cross-sectional area of the blasthole.

To date, the most widely used explosive has been a non-detonator sensitive, TNT-based water gel.

In weak rocks, water gels with lower velocities and densities have performed satisfactorily.

In general, ANFO-type blasting agents are unsuited to VCR on account of their relatively low densities and velocities.

3.4 Crater Testing

In order to determine both the maximum crater volume and optimum depth of burial of the charge, it is necessary to carry out crater tests. The principal considerations in such tests are as follows.

1. Crater tests must be carried out in the same rock and with the same explosive as the production blasts.

2. The diameter of the crater test blasthole should be as large as possible.

3. A series of test blastholes having a wide range of lengths must be fired (to cover a range of depths of burial).

4. Test blastholes should be collared along the centreline of a drive wall. They should be horizontal and normal to the wall.

5. All test charges must have the same weight and should be well tamped in the blasthole.

After firing the test blastholes, the craters are measured and the results are then plotted on a graph as shown in Figure A.1. The optimum crater dimensions (obtained from the graph) are scaled up to give production blast parameters.

3.5 Charging Techniques for Production Blastholes

When charging production blastholes, the following procedures should be adopted.

1. In order to determine the profile of the back, each blasthole is measured with a non-stretch tape attached alternatively to wooden sticks 1.0 m (3.2 ft.) and 0.3 m (0.9 ft.) long. Lower the 1 m (3.2 ft.) stick on the measuring tape until it leaves the bottom of the blasthole. The pull up the tape to determine the distance between the back and the collar of the blasthole. The tape is attached to one end of the stick by a rope having a length exceeding half the stick length and the end of the tape is then connected to the centre of gravity of the stick by a short piece of bell wire (see Figure A.2). To recover the stick, a sharp jerk on the tape will break the bell wire and allow the stick to hang vertically. It is important to note that the measuring of blastholes must be carried out immediately prior to charging. Any previous measurements may be in error as a result of rockfalls during the intervening period.

2. Blastholes can be blocked off with either a solid wooden plug or a wooden split wedge (see Figure A.3). In the former case, the plug is lowered on a non-stretch rope to the design position in the blasthole. This plug is set by dropping (say) 15-22 mm (0.5 in.-9 in.) angular crushed rock on top of it and tensioning it to a steel bar across the collar of the hole. crushed rock is

added until the design bottom of charge is reached. Alternately a short stick (approximately 50 cm (20 in.)) can be attached to the rope below the plug and the plug is then tied off a specified distance above the stick. The plug may be set immediately below the charge or in such a position that stemming may be placed below the charge. The plug is lowered until the stick exits the blasthole bottom; then the rope is tensioned and tied to a steel bar across the blasthole collar.

Where the more expensive split wedges are used, the wedge is set by dropping a heavy rock or retrievable steel bar on top of it. Once set, the distance to the top of the wedge should be measured and stemming added to obtain the design depth to bottom of charge.

3. The explosive charge should be centrally primed. This is achieved by lowering:

   (a) a 400 g cast primer on a 10 g m downline; or

   (b) a cartridge of explosive primed as in (a) above.

4. The charged blasthole should not be measured to determine the position of the top of the charge.

5. The stemming column above the charge should be 12 blasthole diameters long. Shorter columns reduce blast effectiveness. Longer columns tend to encourage freezing of the stemming thereby hindering or even preventing the next charging operation. It is for this latter reason that angular crushed rock should not be used to stem above the charge. Fine sand should be employed.

6. Fire the blast using a short delay technique based upon:

   (a) millisecond delay detonators attached to the downlines at the collars of blastholes; or

   (b) detonating relay connectors inserted into a detonating cord trunkline network.

   The former method is the more preferable.

7. In general, delay number should be allocated such that easiest breaking blastholes fire first. The remaining blasthole are then delayed so as to strip to the voids created by the earliest firing charges (see Figure A.4).

3.6 Practical Considerations

Where some of the blastholes do not break through into the undercut area below (see Figure A.5), only the breakthrough blastholes are charged and fired initially. When these VCR blasts have raised the back sufficiently, non-breakthrough blastholes can be gradually included in the blast design. This is shown by the numbering of backs and the lettering of blast blocks in Figure A.5.

Blasting of the crown (directly beneath the topsill) requires the blasting engineer's best judgement. As an example, let us consider the blasting of crowns which are 1 1/2, 2 and 2 1/2 times the mean upward advance (= x metres).

1. A crown 3x/2 metres (6.5 ft.) thick should not be taken out in two separate blasts, since the first blast would leave a highly-fractured layer only x/2 metres (6.5 ft.) thick which would not be a sufficiently safe surface from which to charge the final blast. This remaining slice should be removed in a single blast, charges of the standard weight being positioned equidistac from each horizontal surface (see Figure A.6).

2. A crown 2x metres (6.5 ft.) thick should be removed in a single blast using two decked but simultaneously-initiated charges in each blasthole (see Figure A.7).

3. A crown 5x/2 metres (6.5 ft.) thick should be taken out in two separate blasts. The first blast removes a slice x metres thick. The remaining 3x/2 metres (6.5 ft.) crown is taken out as shown in Figure A.6.

Differences in ground conditions make it impossible to give specific rules on the distance for which one should use single charges, double charges and double blasts. However, the figures shown in Table A.1 may be used as a general guide.

TABLE 4-A.1
GENERAL GUIDE FOR FINAL CROWN
BLAST DESIGN

Multiple of Mean Vertical Advance	Procedure
1.5	single blast; single symmetrically placed charge (see Figure A.6)
1.6-2.0	single blast; decked charges fired simultaneously (see Figure A.7)
>2.0	two seperate blasts

The speed of drawing the blasted ore must be such as to provide sufficient void for swell. If the volume of the underlying void is too small, the explosion gases are forced to jet out of the blastholes thereby tending to disrupt the collars of these blastholes and hinder subsequent charging operations.

3.7 Blast Records

More than in any other blasting method, VCR requires a meticulous approach in measuring, charging and stemming blastholes. Detailed blast records are an essential component of the VCR method. A typical blast record is shown in Table A.2.

3.8 References

HARRIES, G., 1973. A Mathematical Model of Cratering and Blasting. Proc. Nat. Symp. on Rock Fragmentation (Aust. Geomech. Soc.), Adelaide, p. 41.

HARRIES, G., 1977. The Calculation of the Fragmentation of Rock From Cratering. Proc 15th APCOM Symp., Brisbane, p. 325.

LANG, L.C., 1978. Cratering Theory Evolves into New Underground Mining Technique. Proc. Au. I.M.M.'s Rock Breaking Equipment and Techniques Symp., Nov., p. 115.

LIVINGSTON, C.W., 1956. Fundamental Concepts of Rock Failure. Colo. Sch. Mines Q., Vol. 51, No. 3, p. 1.

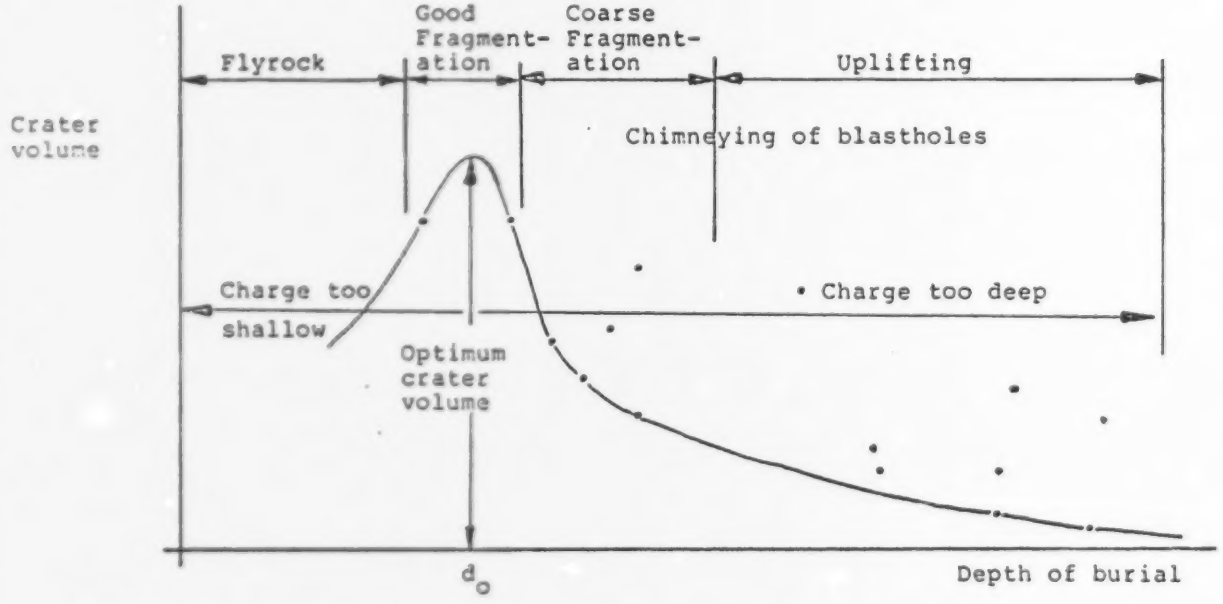
TABLE 4-A.2

A TYPICAL BLAST RECORD

HOLE NO.	HOLD DEPTH BEFORE (M)		DEPTH TO BOTTOM OF CHARGE (M)	DEPTH TO TOP OF CHARGE (M)	CHARGE LENGTH (M)	DEPTH TO CENTRE OF CHARGE (M)	CHARGE WEIGHT (KG)	HOLE DEPTH AFTER (M)		ADVANCE (M)
	1.0M STICK	0.3M STICK						1.0M STICK	0.3M STICK	
1	24.2	24.2	22.2	21.2	1.0	2.5	37.5	20.5	20.4	3.8
2	23.6	23.6	21.6	20.5	1.1	2.55	37.5	20.0	19.8	3.6
3	24.5	24.2	22.5	21.6	0.9	2.45	37.5	20.8	20.8	3.7
4	23.9	23.7	21.9	20.9	1.0	2.5	37.5	20.2	20.1	3.7
5	24.7	24.7	22.8	21.8	1.0	2.4	37.5	21.3	21.2	3.4
6	24.2	24.1	22.1	21.2	0.9	2.55	37.5	20.7	20.7	3.5

TYPICAL GRAPH OF CRATER TEST RESULTS

Figure **A.1**



d_o = optimum depth of burial

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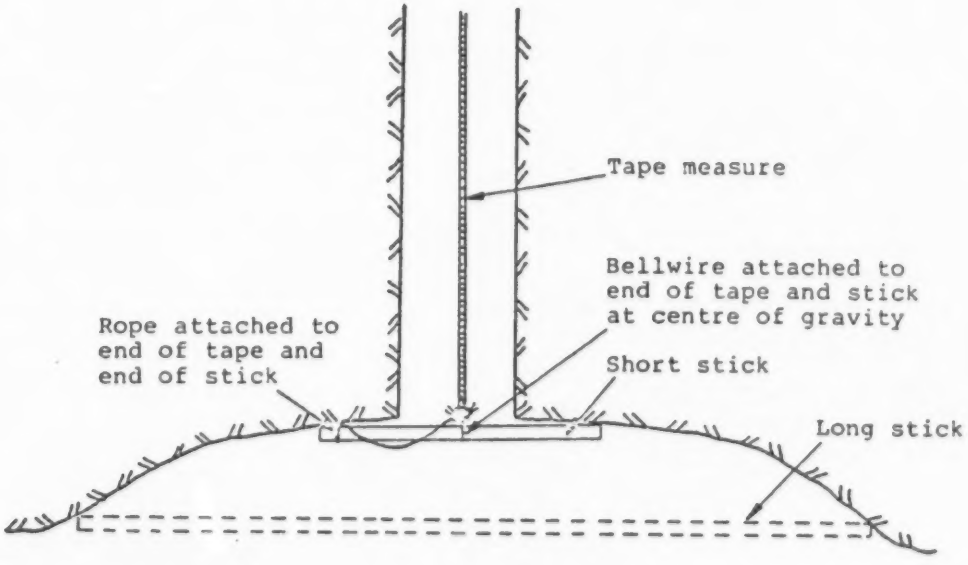
MEASUREMENT OF BLASTHOLE DEPTH

Figure A.2


Tape measure
Bellwire attached to end of tape and stick at centre of gravity
Rope attached to end of tape and end of stick
Short stick
Long stick

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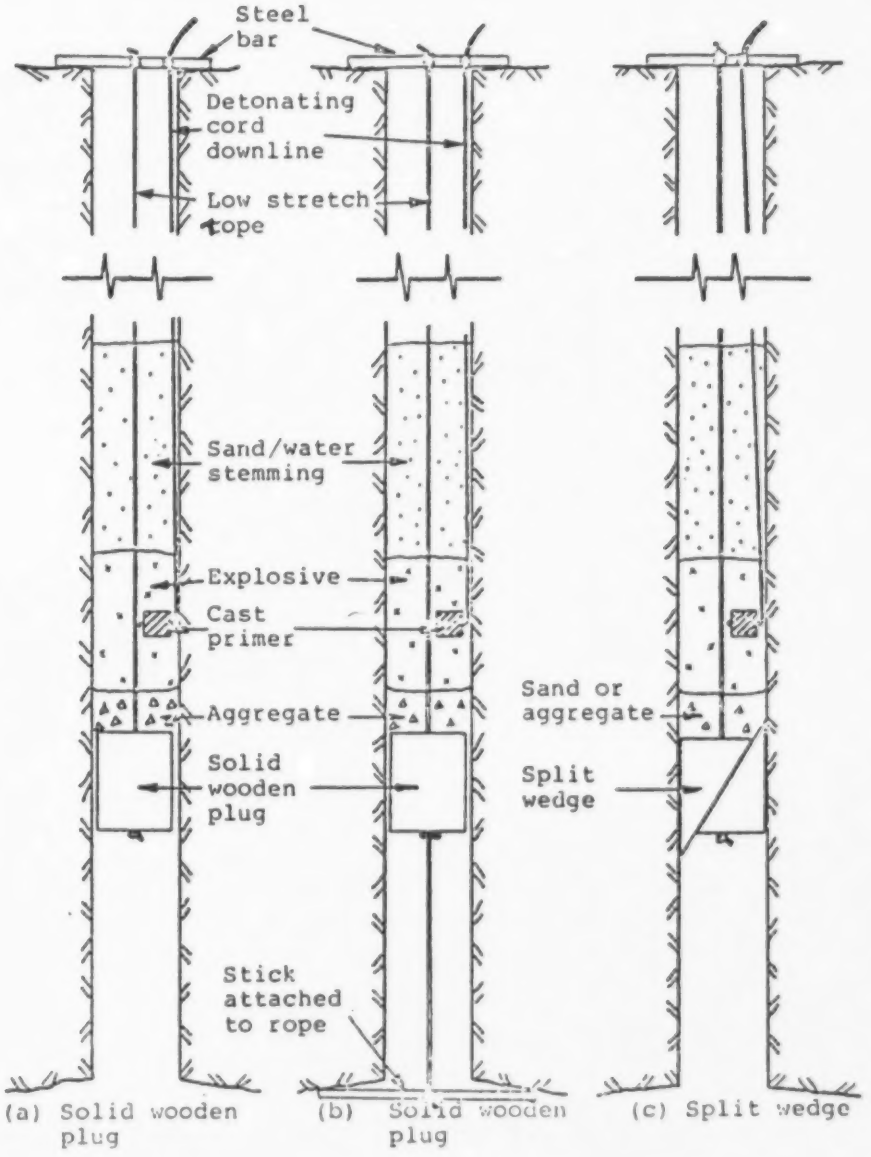
METHODS OF BLOCKING OFF BLASTHOLES

Figure A.3



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A TYPICAL DELAY PATTERN

Figure A.4

Assumed that blastholes on delay No.1 are easiest breaking blastholes.

Delays are millisecond delay detonators attached to detonating cord downline at blasthole collar.

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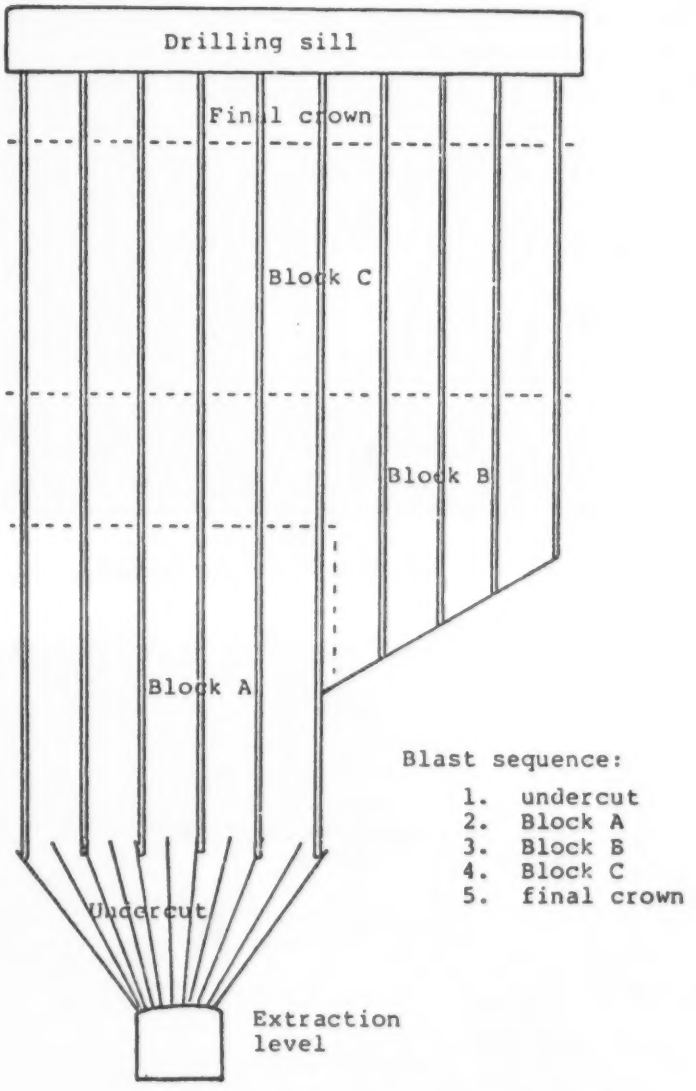
CROSS-SECTION THROUGH STOPE WHERE SOME BLASTHOLES DO NOT BREAKTHROUGH TO UNDERCUT

Figure **A.5**



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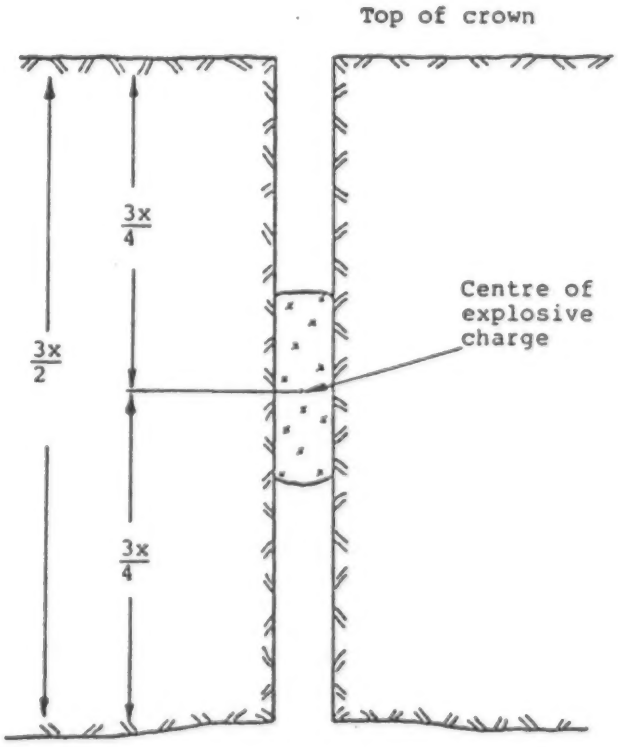
DESIGN OF CROWN BLAST
$\frac{3x}{2}$ METRES THICK

Figure A.6



$\frac{3x}{4} \approx d_o$ = optimum depth of burial

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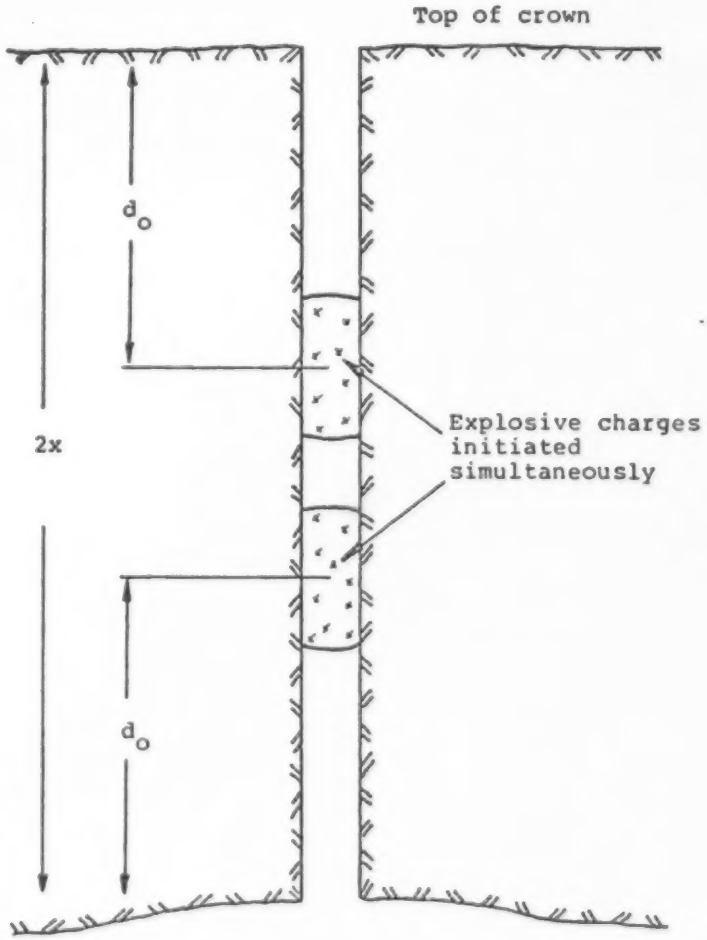
DESIGN OF CROWN BLAST 2x METRES THICK

Figure **A.7**



d_o = optimum depth of burial

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